FOGG

U. S. DEPARTMENT OF THE INTERIOR
BUREAU OF LAND MANAGEMENT

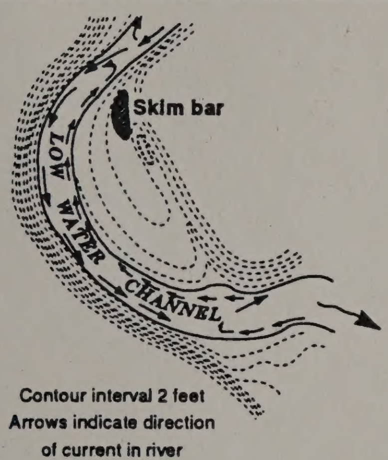
Training Center

Course Number 3000-9

Spring, 1997

Placer Examination Techniques

Part I



Sketch showing the location of flood gold on accretion, or skim bars. (From USGS Bull. 620-L)

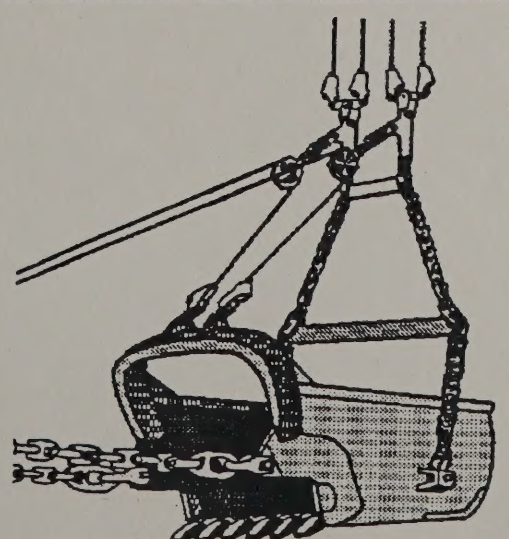
NUGGET EFFECT:
The potentially large value-skewing effect that a single gold particle would have on a small sample as compared to the smaller effect that the same gold particle would have on a much larger sample.

FLPMA

Section 102 (a) (12)

(a) The Congress declares that it is the policy of the United States that –

(12) the public lands be managed in a manner which recognizes the Nation's need for domestic sources of minerals, food, timber, and fiber from the public lands including implementation of the Mining and Minerals Policy Act of 1970 . . . as it pertains to the public lands . . .



Welcome

R 3/22/94

PLACER EXAMINATION TECHNIQUES

BLM Course 3000-9

OBJECTIVES:

1. Identify five of the precious metal placer types and alluvial processes related to each.

2. Identify and discuss major case law applying to placer claims.

3. Apply sampling theory and practices to collect representative samples.

4. Properly collect and process placer samples in the field and laboratory.

5. Accurately calculate sample values and reserves, given data about a property and own calculated values from sampling.

6. Accurately draw a sketch map of a property showing sample locations and reserves.

7. Show preliminary value of ground and cost estimate calculations for the Crystal Hill Mine.

Placer Examination Techniques
BLM Course 3000-09
1997 Final Examination Topics

The 1997 Placer Examination Course will have a final examination. We anticipate that three hours will be available for it. In 1996, most participants completed it in less than two hours. There have been some revisions since 1996, so the time it takes you may be less or more.

Everything in the final examination will be based on the course content, which follows from the course objectives. The instructors will attempt to emphasize topics that will be included in the final examination.

In general, you should expect final examination questions focusing on or including the following topics.

- Sizes, legal aspects, and varieties of placer mining claims.
- Dirtwork calculations.
- Process equipment sizing.
- Swell factors.
- Any and all phases and aspects of placer sampling.
- Designing bedrock drains.
- Resource value calculations.
- Weighted averages.
- Volumetric estimations.

BLM NATIONAL TRAINING CENTER
COURSE EVALUATION
Placer Examination Techniques
Spring, 1997

Your Name________________________ Title________________

Duty Station______________________ GS Grade______________

All sessions attended? Yes No (explain if no)________________

____

General Notes: We really read all the course evaluations. Most of the worthwhile changes in this course have resulted from suggestions and ideas that originated in a course evaluation. The standard questions are used for statistical analysis and we appreciate your responses. The written responses are where we usually find the worthwhile ideas. Please take the time to prepare an honest and reasoned written evaluation. Don't try to complete it in the last few minutes just before class ends, and don't worry about hurting anyone's feelings. All criticism is welcome, but constructive criticism and recommendations will be the most useful. This is your best opportunity to amplify any suggestions or problems that you might have discussed with the coordinator during the course. Your name is optional. But if you make a good suggestion, it is often helpful for us to contact you to discuss it further. As you complete this evaluation, remember that this course has a prerequisite of 3000-1 (prior to 1984), 3000-13 (1985 and later), or equivalent experience with the support of your home office.

Please circle the number that best reflects your reaction. Do not attempt to split a rating because it fouls up our analysis. Where ratings are split, we will assign the next highest number, good or bad.

		Low				High
1.	Clearness of stated course objectives.............	1	2	3	4	5
2.	Overall extent you believe objectives were met.....	1	2	3	4	5
3.	Overall instructional level of the course: too advanced (5), about right (3), too elementary (1)...	1	2	3	4	5
4.	Extent that skills & techniques were new to you....	1	2	3	4	5
5.	Overall applicability of skills/concepts learned to your present job...............................	1	2	3	4	5
6.	Overall applicability of skills/concepts learned to probable future jobs, if for career development....	1	2	3	4	5
7.	Course Length: too long (5), about right (3) too short (1).....................................	1	2	3	4	5

8. Read the following content areas. Put an **X** next to the area you expect to be able to be useful to you on a regular basis while performing your present job and in future jobs as your career develops. Put an **O** next to the area that you don't expect to be useful to you while performing your job.

Present Job		Future Job	Present Job		Future Job
___	Placer Deposit Evaluation.	___	___	Field Mapping.	___
___	Alluvial Placer Mapping Meths	___	___	Mining & Concentration Techs.	___
___	Placer Sampling Procedures.	___	___	Placer Mine Cost Estimating.	___
___	Amalgamation	___	___	Sediment Control	___
___	Reserve Calculation	___	___	Reclamation Planning	___
			___	Concentration Eqpt. Exer.	___

9. This course was designed around the following overall objectives. Please indicate with an X those objectives you feel prepared to perform now, and an O those objectives you feel prepared to perform with initial assistance.

____ Identify five of the precious metal placer deposit types and alluvial processes related to each.
____ Identify and discuss major case law applying to placer claims.
____ Apply sampling theory and practices to collect representative samples.
____ Properly collect and process placer samples in the field and laboratory.
____ Accurately calculate sample values and reserves given data about a property and own calculated values from sampling.
____ Accurately draw a sketch map of a property showing sample locations and reserves.
____ Show preliminary value of ground and cost estimate calculations for the field exercise property.

10. Feedback to the instructors:

Circle the number that best describes how well the instructor performed in each category. Not all categories may be applicable to field instructors. Use blank instructor blocks for instructors added after this evaluation was printed. Do not split ratings as it fouls up our analysis. Where ratings are split, we will assign the next higher number, good or bad.

Instructor: Don Keill.	Poor		Aver.		Excell.
[A] Encouraging and Answering Questions.................	1	2	3	4	5
[B] Using relevant subject matter examples.............	1	2	3	4	5
[C] Involving me (and others) in participative examples.	1	2	3	4	5
[D] Reinforcing critical learning points................	1	2	3	4	5
[E] Using teaching aids (overheads, flipcharts, videos, handouts, and equipment)...................	1	2	3	4	5
[F] Pace of presentation: 1 = too slow, 5 = too fast....	1	2	3	4	5

Instructor: Bob Lewis.	Poor		Aver.		Excell.
[A] Encouraging and Answering Questions.................	1	2	3	4	5
[B] Using relevant subject matter examples.............	1	2	3	4	5
[C] Involving me (and others) in participative examples.	1	2	3	4	5
[D] Reinforcing critical learning points................	1	2	3	4	5
[E] Using teaching aids (overheads, flipcharts, videos, handouts, and equipment)...................	1	2	3	4	5
[F] Pace of presentation: 1 = too slow, 5 = too fast....	1	2	3	4	5

Instructor: Mitchell Leverette	Poor		Aver.		Excell.
[A] Encouraging and Answering Questions.................	1	2	3	4	5
[B] Using relevant subject matter examples.............	1	2	3	4	5
[C] Involving me (and others) in participative examples.	1	2	3	4	5
[D] Reinforcing critical learning points................	1	2	3	4	5
[E] Using teaching aids (equipment, hands-on assistance, etc.)................................	1	2	3	4	5
[F] Pace of presentation: 1 = too slow, 5 = too fast....	1	2	3	4	5

10. Feedback to Instructors, continued

Instructor: Robin McCulloch (Equipment & amalgamation).	Poor		Aver.		Excell.
[A] Encouraging and Answering Questions.................	1	2	3	4	5
[B] Using relevant subject matter examples.............	1	2	3	4	5
[C] Involving me (and others) in participative examples.	1	2	3	4	5
[D] Reinforcing critical learning points................	1	2	3	4	5
[E] Using teaching aids (overheads, flipcharts, slides, handouts, hands on assistance, etc.)........	1	2	3	4	5
[F] Pace of presentation: 1 = too slow, 5 = too fast....	1	2	3	4	5

Instructor: Matt Shumaker (Equipment & bad jokes)	Poor		Aver.		Excell.
[A] Encouraging and Answering Questions.................	1	2	3	4	5
[B] Using relevant subject matter examples.............	1	2	3	4	5
[C] Involving me (and others) in participative examples.	1	2	3	4	5
[D] Reinforcing critical learning points................	1	2	3	4	5
[E] Using teaching aids (overheads, flipcharts, videos, handouts, when appropriate).................	1	2	3	4	5
[F] Pace of presentation: 1 = too slow, 5 = too fast....	1	2	3	4	5

Instructor: ____.	Poor		Aver.		Excell.
[A] Encouraging and Answering Questions.................	1	2	3	4	5
[B] Using relevant subject matter examples.............	1	2	3	4	5
[C] Involving me (and others) in participative examples.	1	2	3	4	5
[D] Reinforcing critical learning points................	1	2	3	4	5
[E] Using teaching aids (overheads, flipcharts, videos, handouts, as appropriate)..................	1	2	3	4	5
[F] Pace of presentation: 1 = too slow, 5 = too fast....	1	2	3	4	5

Instructor: ____.	Poor		Aver.		Excell.
[A] Encouraging and Answering Questions.................	1	2	3	4	5
[B] Using relevant subject matter examples.............	1	2	3	4	5
[C] Involving me (and others) in participative examples.	1	2	3	4	5
[D] Reinforcing critical learning points................	1	2	3	4	5
[E] Using teaching aids (overheads, flipcharts, videos, handouts, as appropriate)..................	1	2	3	4	5
[F] Pace of presentation: 1 = too slow, 5 = too fast....	1	2	3	4	5

11. Why did you attend this course? Check all that apply by numbering them in priority order.

___________Improve performance in my present position.
___________Learn critical new skills.
___________Maintain performance in my present position.
___________Meet mineral examiner certification training requirements.
___________Learn new information for a new position of responsibility.
___________Other (describe)____

12. Were the course topics sequenced logically? Yes No
If not, how should they be sequenced?

____

____

13. Which subjects covered in the course would you regard as the most valuable to your job? (briefly explain why)

14. Which subjects covered in the course would you regard as the least valuable to your job? (briefly explain why)

15. Did the information presented in this course meet your expectations and needs? If not, why?

16. Describe any changes you will make in your job because of this course.

17. What topics should have been added or given more emphasis?

18. What topics should have been deleted or given less emphasis?

19. If you were to redesign this course, what changes would you make and why?

20. What parts of the notebooks should have been omitted?

____

____

____

21. Comments on transportation and lodging?

____

____

____

22. Do you believe that NTC should conduct a course on watershed and wetland rehabilitation, which would include placer mine reclamation? Yes? No? Explain.

____

____

____

23. We hope to redesign this course to reduce the amount of time that students spend in travel status. Other than going to a 24 hour training day, a potential approach is through distance learning, which can encompass many training methods. These include correspondence courses, advance assignments, interactive CD-ROM, real-time satellite broadcasts, etc.

A. What parts of the segment you've recently completed do you think would lend themselves to distance learning methods? Would you be able to devote the necessary time at your home duty station? Be realistic, and include your home office work load in your thoughts.

____

____

____

B. What delivery methods (i.e., satellite broadcasts, self study, video, audio tape, CD ROM, etc.) would you be most responsive to? Costs are also a consideration, and there are many hidden costs. For example, satellite time at $900-plus per hour plus equipment costs can be cost effective with a large enough audience.

____

____

____

If you have additional comments (such as to amplify answers from the standard questions at the beginning of this form) or suggestions, please use the back of this page or additional sheets of paper. It's a good idea to reinforce any verbal suggestions that you might have made to the coordinators in writing here.

Mining Law
Overview

EMPTY SECTION

Keill's Lesson
Plan Outline

ESSON PLAN TWO: PLACER DEPOSITS

JBJECTIVE: LIST AND DISCUSS MAJOR CASE LAW APPLYING TO PLACER CLAIMS

TIME	SUBJECT	METHOD	ACTIONS
2hr.			

I. Introduction

Case law is a very complex subject as you are aware and a brief discussion would take a week or more. We will highlight the major laws or decisions that afffect placer operations only.

~~120~~ 30 min. II. Case Law Overview and Summary

A. What is a profitable mine? a paying mine? — question class — write answers on board

a. Castle vs. Womble, 19 LD 455, 1894

B. How much is the quantity and at what quality?

a. large yardage/ low grade?
b. low yardage/ high grade?
c. effects on cost estimates?
d. what environmental costs apply?
Murdock, 65 IBLA 239 (1982)
McKenzie, 29 IBLA 270 (1977)

C. What constitutes a discovery?

a. is there a difference between validity or a patent application?
b. quality of mineral or quantity?
c. how much does the examiner do to verify the discovery, dig a lot or collect from open exposures only?
(IBLA Anderson decision)
d. price fluctuations or spot price?
(re. Pacific Coast Molybdenum)
e. best intentions of the claimant?
f. amount of equipment on site or owned by the claimant or partners?
g. suitability of the mining method or proposed method or one developed by the field examiner?

BREAK 15 MINUTES AT END OF 60 MINUTES OR AS NECESSARY

ME	SUBJECT	METHOD	ACTION

15 min D. Claim size and the 10 acre Rule

a. acres in size per locator, 160 acres maximum with 8 locators
b. state law may further limit size
c. are to be in cardinal directions unless located as a gulch placer
d. one discovery holds each claim
e. each 10 acres must be mineral in character
f. claim must be divided perpendicular to long dimension as equal as possible
g. must sample each 10 acres unless geology is continuous and consistent.
h. examiner may sample more than once per 10 ac. if claimant requests or examiner determines a need.

15 min E. Mineral Character

a. definition of, see Southern Pacific Co. 71 ID 224,233,(1964)
b. evidence supporting mineral character Southern Pacific Co. 251 US 1,14 (1919) and other cases
c. discovery vs. mineral character
d. mineral character and 10 ac. Rule
e. mineral character and quantity of mineral required to justify cost of extraction

Example Problem:

20 min. A 20 acre placer claim has a clear discovery on one 10 acre portion. Two samples were collected from the second 10 acre part. One sample had values above the mining cost for the property and the other was below the operating cost.

Is the second 10 acres mineral in character? WHY?

10 min. QUESTIONS AND DISCUSSION

END LESSON TWO

BREAK 15 MINUTES AT END

LESSON PLAN ONE

COURSE TITLE: PLACER DEPOSITS

TOTAL TIME: 2 HOURS

OBJECTIVE: IDENTIFY FIVE PLACER TYPES AND THE ALLUVIAL PROCESS RELATED TO EACH

TRAINING AIDS:

EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
40 Min	**I. Introduction** Placer: a) deposits of minerals that are not in placer; b) deposits of detrital materials containing valuable concentrations of minerals.	Lecture	
	Prerequisite: 1. valuable heavy mineral 2. resistant to weathering and abrasion 3. released from parent material 4. concentration in zones or deposits		question class list on board
	Formation: Each placer is unique from all others in one or more ways. The richness and size of each is more dependant on abundant source materials and favorable concentration conditions rather than richness of the source. of Sampling		emphasize
	Sources: 1. lode-veins or disseminated 2. erosion of paleoplacer source 3. reworking of glacial material 4. magmatic segregating		
	Weathering and erosion processes: 1. ground water (solution) 2. temperature differential 3. biological and chemical 4. eolian (wind) 5. normal stream flow equilibrium 6. bed scour and agitation 7. storm/flood transportation 8. paleontopography vs. present		

	Concentration: 1. bedrock (fractured, jointed, clay weathered rocks = best; smooth, hard, flat laying = poor) 2. false bedrock layers 3. paystreaks/ sinuous enrichments 4. flood concentrations (many thin layers) **Preservation:** 1. abandonment of channel 2. rapid downcutting 3. regional uplift 4. rapid burial (avalanches, landslides, slumping, volcanic flows, flood events)		draw block diagram
1 hr	**II. Types of Placer Deposits**	Lecture	OH
	1. **Residual** - near lode, crystals, wire, no transport, shallow depth, very rich, pocket concentrations		slide A slide 1 slide 2 slide 3
	2. **Eluvial** - creep transition between residual and colluvial, subangular xtls, wire, transport: 10s of feet, depth ,20 feet, very rich horizontal deposits.		slide 4 slide 5
	3. **Alluvial** - sorted, transported and deposited by water. a. **gulch** - single cycle of erosion transported and deposition, poorly sorted, discontinuous paystreaks, high boulder content, ephemeral, steep streams, well conc. on bedrock, narrow, coarse nuggets, subangular.		slide 6 slide 7
	b. **creek (stream)** - sorted or resorted, rounded gravels, moderate grade, defined paystreaks, narrow to wide, concentration on bedrock, may be vertically graded,false bedrock, subangular to rounded nuggets, moderate boulder content		slide 8 slide 9 slide 10 slide 11
	c. **river** - resorted, flattened and rounded gravels, low gradient, long continuous paystreaks, disseminated vertically, false bedrock, low grade, conc. on upstream end of bars, fine, flattened flakes and grains of gold mined by dredges or large scale methods.		slide 12 slide 13
	d. **gravel plain** - large deposits, flood or deltaic deposits, disseminated		slide 14

	throughout, fine sized gold, flattened wide discontinuous paystreaks, may not be amenable to mining. Yuba River	
	e. **flood deposits** - high velocity events very poor sorting, shallow depth, very fine gold size (5000 pr 1 mg), skim bars, usually not minable, Snake River deposits.	slide 15
	f. **bench** - remnants of earlier alluvial processes, rapid downcutting, uplift, usually richer than present stream deposits, coarse, angular to rounded gold, discontinuous paystreaks, false bedrock. Many present operations mining benches.	slide 16 slide 17
	g. **buried** - rapid increase in bedload volcanic flows, ie. "Tertiary gravels" isolated from present stream or topography, cemented gravels, well weathered.	slide 18
	h. **desert** - aka. "bajada", high velocity storm events, bedload scouring, very angular gravels, poorly sorted, surface enrichment due to wind, erratic distribution generally, conc. on bedrock near toe of mountains.	slide 20
	i. **beach** - wave and current breakdown and conc., very fine close size particles, rich in heavy minerals, erratic, lenticulas shape, no channels, thin, flat gold.	slide 21 slide 22 slide 23
	j. **glacial** - erratic distribution, high boulder content, subangular nuggets, smooth bedrock, low value, variable paystreaks.	slide 24
10 mins.	QUESTION AND DISCUSSION	
	How Thorough Should an Examination Be?	OH
	END LESSON ONE	
	BREAK - 15 MINUTES	

LESSON PLAN THREE

COURSE TITLE: PLACER DEPOSITS

TOTAL TIME: 1-2 HOURS

OBJECTIVE: APPLY SAMPLING THEORY AND PRACTICES TO COLLECT A REPRESENTATIVE SAMPLE

TRAINING AIDS:

EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
1-2 hrs	**I. Introduction** "Placer samples provide limited information and the examiner must use the powers of deduction and experience based upon judgment to correctly evaluate placer properties... rather than a rigid application of formulas and procedures." --John Wells, Placer Examination, Tech. Bulletin #4		
90 mins	**II. Sampling Theory** Developed over the last 20 years through the work of Pierre Gy and Franis Pitard and others. They have taken the intuitive and sometimes biased approach to sampling and have developed a theoretical mathematic approach and are developing the practical applications of the theory to field practices. This is giving new ways of collecting samples, collecting more representative samples and reproducible results. Main concern in sampling theory and practice is obtaining a representative and accurate sample.		

Sampling is a selecting process.		OH 1
Two categories:		
<u>Probabilistic</u>	<u>Non-propabilistic</u>	
all elements submitted for selection	not prime reason for selecting	OH 2
have given probability of being selected	deterministic (grab sample) purposive (chip or shovel)	
work will on compact ore bodies	typical of particulate materials	
well studied by geostaticians	great difficulties to implement correct sampling	

4. Non-probabilistic sampling: -no theoretical approach -sampling errors are large enough to deprive sample of any practical value -still taught and used in commercial sampling, mining evaluations, process control and pilot plants		OH 3 OH 4
a. Purposive sampling (chip or shovel) -operator selects the sample from the most accessible or easiest to collect. -accuracy depends on the operator's choice -operator is not uniform or consistent -human nature is biased depending upon what is trying to be proven. -generally the sample is not accurate representation of material.		OH 5 OH 6
b. Deterministic sampling (grab auger, drill) -collecting of increments from the most easily accessible part of the lot with hands, shovels, scoops, splitters. -because one recognizes there are variances within the lot the person collecting the sample tries to take as many increments as is practical. -a large portion of the lot is systematically kept away from the sampling tool or procedure: * bottom of a truck or rail car * bottom of a drum * center of a conveyor belt * center of a pile or loader bucket -use of probes, augers, drills, etc. only amount to improve grab sampling tech.	OH 7	

	-idea is to extract a complete and undisturbed column representing the lot at a predetermined point. -works only when the point is selected in a manner that is probabilistic, which is rarely done. -the sample point is usually pre-selected: * drilling the center of a barrel, * drilling a diagonal along a bag, * drilling from the top of a truck, * drilling from the top of a pile.... -if these locations could be probabilistic, which is never the case, the results obtained never are correct. -conclusion: grab sampling is not accurate because some portions of the lot have a zero chance of being selected and the probability of selecting between units making up the lot cannot be maintained constant. -doesn't conform to the sampling theory as there is no way to keep track of all the various sampling errors. -the sampling will always be biased but to an unknown extent. -this type of sampling gives indication of the value of ground or lot, not necessarily the true value.	class discussion
10 mins	QUESTIONS AND DISCUSSION	
	Conclusions:	
	Samples seldom easy to obtain	OH 8
	Generally samples should be large	OH 9
	Sampling must follow a probabilistic system	
	Some errors can be controlled, others cannot	OH 10
	Sample results may only give indication of value of the ground	
	Distribution of values makes accurate sampling difficult	OH 11
	High unit values magnify all other errors made in sampling	OH 12
		Handout 3-1 Handout 3-2
	END LESSON THREE	
	BREAK - 15 MINUTES	

LESSON PLAN FOUR

COURSE TITLE: PLACER CLAIM EVALUATION

TOTAL TIME:

OBJECTIVE: THE SUCCESSFUL STUDENT WILL BE ABLE TO:
PREVENT OR MINIMIZE SALTING PRACTICES WHEN COLLECTING SAMPLES

TRAINING AIDS:

EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
1 hr	Sampling Errors and the Reliability of Sampling **A. Errors common to all Placer Environment** Incorrect sample dimensions Faulty analytical procedures Salting Carelessness Inaccurate measurements Drilling into a hard bottom Use of drilling correction factors Inaccurate logging Unsuitable drilling equipment Uneven distribution of values Not allowing for dilution and batters Ignorance Errors in Continental Placer Sampling Splitting samples Irregular-shaped pits and channels Errors in Transitional Placer Sampling Grain counting Failure to recognize changes in lithology Errors in Marine Placer Sampling *Source: Alluvial Mining, By Eoin Macdonald, 1983, pages 227-233.	GP	OH 1

B. **Sampling Considerations**	Lecture	OH 4
1. Adequacy of size to obtain representive sample 2. Uniform width and depth of channel from top to bottom. 3. Adequacy of sample to character of the ground. 4. Spillage of sample while collecting. 5. Spillage of sample while processing. 6. Water washing bedrock just prior to collecting sample. 7. Erratic collection of samples from backhoe bucket -- should sample every bucket.		
8. Drill mast not vertical. 9. Natural and introduced oils in the sample: avoid!! 10. Soil running into drill hole: dilution 11. Excessive reaming or bailing of hole prior to sample collection 12. Inadvertent release of core holder (worn parts, carelessness, etc.) 13. Overfilling collection containers and spilling sample. 14. Dirty equipment of both applicant and BLM. Clean all tools and machines prior to running each sample		OH 5 OH 6-16 Add Info 3-1

LESSON PLAN FIVE

COURSE TITLE: PLACER CLAIM EVALUATION

TOTAL TIME: 1 1/2 HOURS

OBJECTIVE: THE SUCCESSFUL STUDENT WILL BE ABLE TO:
SELECT AN APPROPRIATE SAMPLE METHOD AND LIST ADVANTAGES AND DISADVANTAGES OF THE METHOD.

TRAINING AIDS:

EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
1 1/2 hr	I. SAMPLE METHODS	Lecture	OH 1 Line 4
30-45 min	1. hand dug pits, trenches or shallow shafts	Discuss	slide 25
	a. advantages:		
	1. good for shallow ground <8'		
	2. dug by unskilled workers		slide 26
	3. may permit resampling		
	4. good estimation of boulders and character at the deposit		slide 27
	5. no specialized equipment required		slide 28
	6. good for remote locations		
	b. disadvantages:		
	1. ground must be dry, non-caving		
	2. must have cheap labor		
	3. time consuming		slide 29
	4. not practical for deep ground		
	2. machine dug pits and trenches		slide 30
	a. advantages:		
	1. quick and easy to change sample sites		
	2. large samples possible		slide 31
	3. handle variety of placer ground		slide 32
	4. channel or bulk sampling possible		slide 33
	5. character of ground observable		
	6. wet conditions handled		slide 34
	7. bedrock easily reached and cleaned (generally)		slide 35

	b. disadvantages:	
	1. costly to operate	
	2. require skilled operator	slide 36
	3. limited digging depth for backhoes	
	4. may disturb sample (unintentional salting)	
	3. Shafts - hand or mechanically dug	
	a. advantages:	
	1. good view of ground conditions	slide 37
	2. good samples obtained	slide 38
	3. well placed shafts may preclude other sample programs - can begin mining	slide 39
	4. mechanized, relatively fast sinking rates	
	b. disadvantages:	
	1. high cost	slide 40
	2. skilled labor required	
	3. difficult to obtain accurate	
	4. can't be used in wet, running ground	
	5. generally small diameter no room to work, 24-36" diameter samples	slide 41
	4. drills - many types used - all have advantages	slide 42
	a. Churn drill-oldest, most accurate, studies, reliable, drill all ground conditions, 100% recovery possible	slide 43 slide 44
	b. rotary-fast, drill medium depths, mixes sample, unknown recovery, rates vary with driller, 40-75%, machine	slide 45 slide 46
	c. auger-moderate speed, shallow ground, mixes sample, unknown recovery rates	
	d. reverse circulation-fast, expensive recovery rates - approach 100%, drill deep ground	slide 47 slide 48
	e. resonant drill-fast, expensive varied recovery from zero to 100%, drill deep ground	slide 49
	f. Banka/Empire drill-hand powered, shallow ground, all drilling requires correction factors for recovery in order to accurately place a value on placer ground, plus or minus corrections Radford factors commonly applied to churn drills: must determine if a	

factor was used when evaluating old drill logs. Examples: churn drilling with ground run-ins, rotary air return-damp ground		
5. Dredges	Covered by others	
a. suction dredges - value per time, not yards processed - need well sorted material, special apparatus required	Video/demonstration	
II. Sample Site selection		
a. drill holes - ask claimant		slide 50
1. fence - row of holes perpendicular to paystreak		
2. random - best site selected by geology, etc. factors		
3. grid pattern - define limits on sides of pay		
b. existing exposures - intentional and unintentional, salting may be a concern		slide 51 slide 52
c. geophysical anomalies - aerial, magnetic, electrical, seismic		slide 53 slide 54 slide 55
d. intuition or "hunch" witching		slide 56-58
Record sample data in field:		
1. field guide and checklist for placer examinations	See field	OH 1 checklist
2. field notes "rite in rain" notebook - estimated quantity of Au. recovered		OH 2 OH 3
3. BLM Form 3842-1, 3842-2, 3842-3		
4. photographs, number with index card in picture		OH 4
5. double bag concentrates when transporting		
BREAK - 15 MINUTES		
III. Field Sampling - may use a variety of methods all on one property, must be familiar with them all		slide 59
1. Channel samples: must		slide 60
a. be uniform in width and depth		
b. collect representative amounts of all size fractions		
c. be collected without being salted, intentional		
d. be taken in ground that does not cave		slide 61
e. be collected quickly if water flowing into sample area		slide 62

2. Backhoe bucket samples		slide 63
a. must take equal portions of bucket or consistently from same part of the bucket to avoid salting:unintentional		slide 64 slide 65
b. collect representative sample of all material		slide 66
c. may be collected from wet or dry conditions		slide 67-68
3. Bedrock samples		
a. collected as a "check" sample from the actual mining area		slide 69
b. hard to determine exact value per yard		slide 70
c. must have dry conditions		
4. Bulk samples		slide 71
a. time consuming to process		
b. most valid value of ground obtained		slide 72-73
c. require assistance of many employees		slide 74-75
5. Drill samples		
Generally collected in a dump or mud box from which the sample flows into a bucket and fines and slimes are floated off. The volume is measured in the bucket and used in the calculations for value per yard. Weights also kept of the sample as a check		slide 76 slide 77
IV. Field Processing:		
Make sure all equipment is present and in good working order. Valves working, engine oil and filters - clean, tool kits and spare parts are along. May use any or combination of:	Exercise/Discussion on location	
A. Large volume sampling		
1. panning: use 12" or 14" gold pans and pan in a 16" safety pan inside a shallow flat tub, tiring, light weight		slide 78 how not to slide 79
2. rocker box: use smooth motion; don't splash water in; make sure there are leaks along the riffles or side boards, tiring, light weight		slide 80 slide 81
3. E-Z Panner: vibrating screens oscillating riffles good on easy washing gravels, poor in clay rich Removable riffle drawer. Good for other minerals (e.g. tin, nuggets, garnets, etc., heavy weight)		slide 82 slide 83 slide 84
4. Gold Saver: trommel screen, oscillating riffles, good in clayey		slide 85 slide 86

gravels and easy washing gravels, easy to modify spray bar to enhance washing ability, difficult to remove riffle tray, heavy		slide 87
5. Elevated sluice box: convertible to suction dredge or for use on dry land easy to clean up; poor washing of clayey material, simple to operate, light weight.		slide 88 slide 90
B. Small volume samples 1. gold pan. See No 1 above 2. Micro sluice: small size, very light weight, indoor use recommended, slow feed, sensitive to changes of slope or water flow.		slide 91
3. Gold Wheels - bulky, medium weight require semi-controlled environment needed, sensitive to changes of slope or water flow.		slide 92
C. Bag all concentrates in "ziplock" bags with any free gold extracted and placed in a 4 dram vial. Mark bags with water-proof marker and place in another bag for protection.		slide 93 slide 94
D. Secure all samples in a location not accessible to others.		
END		

LESSON PLAN FIVE-A

COURSE TITLE: PLACER CLAIM EVALUATION

TOTAL TIME:

OBJECTIVE: THE SUCCESSFUL STUDENT WILL BE ABLE TO:
LIST EFFICIENCIES OF VARIOUS RECOVERY DEVICES AND LIST WAYS TO INCREASE SAMPLE RECOVERY EFFICIENCY

TRAINING AIDS:

EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
30 mins	V. Efficiency of the Equipment and Sampling Methods		OH 5
	A. Percent of Recovery by Type of Equipment		
	Hand Shovel - 100 unless large run		
	Suction Dredge - 70-80		
	Backhoe Buckets - 90-100		
	Auger - 20-50		
	Banka Drill - 0-100		
	Hand Churn Drill - 90-100 needs correction factors		
	Light Churn Drill - 90-100 needs correction factors		
	Heavy Churn Drill - 90-100 needs correction correction factors		
	Hand Dug Cassions - 80-100 needs correction factors		
	Machine Dug Cassions - 80-100		
	Drill-Percussion - 80-95 needs correction factors		
	Drill-Rotary - 40-75 needs correction factors		
	Drill-Reverse Circ. - 90-100 needs correction factors		
	Drill-Vibratory - 80-90 needs correction factors		
	Drill-Resonant - 95-100 needs correction factors		

B. Efficiencies of Sampling Devices — OH 6

Gold Pan - nuggets to 100 microns
98-100%

Sluice Box:

Hungarian riffles - nuggets to 200 mesh
70-80%

Chinese riffles - nuggets to 200 mesh
80-90%

Regular riffles - nuggets to 100 mesh
50-90%

Expanded Metal - -28 mesh to 400 mesh
90-98%

Rocker Box - nuggets to 200 mesh
75-90%

Gold Saver - (trommel) -1/4" to 100 mesh
85-92%

E-Z Panner - (vibrating screen) -3/4" to 200 mesh
85-96%

Air Bellows - (dry washers) -1/4" to 100 mesh
70-85%

Spiral Wheel - (goldhound) -20 to 250 mesh
80-90%

C. How to Increase Efficiency — OH 7

Sluice Box: increase grade from
1 1/2":1' to 2 1/4":1'

Increase water percent of feed slurry
90% + (i.e. decrease density of feed)

Add a wetting agent: soap or shampoo

Add an acid (dilute) to feed water

Classify feed by screening

D. Alaska & Canadian Studies of riffle and sluice efficiencies — OH 8 Add. Info
OH 9
OH 10 Add. Info

BREAK

LESSON PLAN SIX

COURSE TITLE: PLACER DEPOSITS

TOTAL TIME: 1 1/2 HR

OBJECTIVE: ACCURATELY CALCULATE SAMPLE VALUES AND RESERVES, GIVEN DATA ABOUT A PROPERTY AND USING OWN CALCULATED VALUES.

TRAINING AIDS:

EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
5 mins	I. Introduction By convention, most placer samples are reported in grams or milligrams per unit of measure, whether this is yards, bedrock feet or foot of drive with a drill. Also, almost all calculations for a placer property relate to the bank run volume. This is very important to remember, as some sample values may be from a loose measure sample lot, i.e. a pile or pilot plant run. Other samples on the property may be measured or from a known volume of bank material. The most common means of measuring sample size are: 1. pan measurements, approximate average: 12" = 400 per yard 14" = 250 16" = 180 2. weight measurements: must know what type of rock is present then, look up weight per yard from excavation tables or, weigh a typical sample in a 1 cubic foot box. 3. volume measurements: measure actual size of excavated hole, pour drill cutting into a calibrated bucket, or pour a known amount of water down a drill hole and calculate the volume based upon depth differences.		Add. Info 6-1

	NOTE: All values reported are actual. recovered milligrams of gold and fineness is not factored into the calculation. In placer evaluation, fineness is a factor only when the product is marketed. There may be a recovery factor for the processing equipment that is considered when the reserves and mining costs are calculated.	OH
	CAUTION: If you calculate $/yard from your sample data, do not use the fineness again to calculate recovered milligrams in an ore block. It is better to calculate all values in one unit or the other ($/yard\3 or mg/yd\3) and then convert any needed values at the end of the calculations.	
10 min	VALUE CALCULATIONS	
	See page V-7 in Mineral Examiners Handbook	OH 1,2,3
	ORE RESERVE CALCULATIONS	
	See pages V-1 - 6 in Mineral Examiners Handbook	
10 mins	1. Block Method	OH
	aka. average end area method. works well with spaced holes or samples in a line across the width of the deposit. Method is fast and accurate for uniform shaped deposits where the whole area will be mined.	
15 mins	2. Triangle Method	
	Works well in irregular shaped deposits, in that areas of lower value may be excluded or averaged with higher valued blocks to even out the mining grade. Also averages out erratic values of samples in that at least one high value must be tied to one low value to form the corners of each triangle. The triangles cannot be formed by only high or low value holes.	OH

mins	3. Polygon Method Works well in irregular shaped deposits, in that areas of lower value may be excluded or averaged with higher valued blocks to even out the mining grade. Also averages out the erratic values of samples in that the sample value's area of influence is the distance half way to the adjacent hole or sample. A disadvantage is that either the sample site's area of influence must by measured in the field accurately or drawn on a map accurately so that a planimeter can be used to calculate the area of influence.	OH
15 mins	4. Kriging A geostatistical grade interpolation method that calculates the grade of a block as a linear combination of the grades of nearest samples. Gives good estimates where the sampling grid is very irregular and where the continuity of the mineralization is different in alternate directions. Through the use of variograms models, kriging develops optimal weights to be applied to each sample in the vicinity of the block being estimated. Kriging utilizes three dimensional locations of samples and works best when there is irregular drill patterns or highly anisotropic ore bodies, in particular, disseminated lode gold or copper properties.	OH
10 mins	QUESTIONS AND DISCUSSION END LESSON SIX BREAK - 15 MINUTES	

LESSON PLAN SEVEN

COURSE TITLE: PLACER DEPOSITS

TOTAL TIME: 8 HRS

OBJECTIVE: ACCURATELY DRAW A SKETCH MAP OF A PROPERTY SHOWING SAMPLE LOCATIONS WORKING AND RESERVES.

TRAINING AIDS:

EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
	I. Introduction		
	Crystal Hill mine and Malakoff Diggings		
15 mins	Divide the students into work groups. Describe general outline of the work area and claim corners.		
	Describe what requirements are expected in the mapping exercise: workings surficial geology gravel reserves by type sample locations sketches of sample site or photos		
4 hrs	Allow students to work at own pace and complete mapping exercise		
	Walk around to each group and monitor activities and answer any questions or provide assistance in completing the exercise.		
1 hr	LUNCH BREAK		
	Informal discussions of progress and problems that are being encountered.		
	Describe the sample site selection and evaluation exercise for next lesson: sampling.		
	QUESTIONS AND DISCUSSION		
	What was seen?		

	What type of alluvial processes are present? What type of sample program is indicated? Mining methods practical and or appropriate? What questions do you have to ask the claimant about the operation? AFTER LUNCH	
3 hrs	Continue Field Mapping and sample site selection BREAK AROUND 4:00 pm	

LESSON PLAN EIGHT

COURSE TITLE: PLACER DEPOSITS

TOTAL TIME: 8 1/2 HOURS

OBJECTIVE: PROPERLY COLLECT AND PROCESS SAMPLES IN THE FIELD.

TRAINING AIDS:

EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
	I. Introduction		
	Return to mine and review previous days lesson.		
25 mins	Have students remain in previous work groups Describe general operation of the Placer Trailer and on-board equipment.		
	Describe what requirements are expected in the sampling exercise: channel samples micro-geology of sample site sufficient sized sample sample locations sketches of sample site or photos Proper processing in Gold Saver and Prospector II		
4 hrs	Allow students to work at own pace and do sampling exercise. Also, continue mapping of the property.		
	Walk around to each group and monitor activities and answer any questions or provide assistance in completing the exercise.		
1 hr	LUNCH BREAK		
	Informal discussions of progress and problems that are being encountered.		
	Describe the sample site selection and evaluation exercise for next lesson: sample analysis		

AFTER LUNCH

[illegible] hrs CONTINUE: field mapping and sample collecting

Demonstrate portable seismograph and allow students to use if they desire.

Have students rotate through as they finish the sample collecting.

Demonstrate operation of the Gold Saver and Knudsen Bowl

Describe proper care and operation of the trailer and related equipment.

Have student groups begin processing of their collected samples. Remainder of groups may watch operation or continue mapping or sample site evaluation.

BREAK AROUND 4:00pm

QUESTIONS AND DISCUSSION

What was seen?

What type of equipment is preferred?

Chain of custody?

Prevention of salting of samples, intentional and unintentional?

What type of sample program is indicated?

Mining method costs, value of the ground and reserves?

What additional questions do you have to ask the claimant about the operation?

END OF FIELD EXERCISE.

LESSON PLAN NINE

COURSE TITLE: PLACER CLAIM EVALUATION

TOTAL TIME:

OBJECTIVE: THE SUCCESSFUL STUDENT WILL BE ABLE TO:
PROCESS PLACER CONCENTRATES WITH THE AVAILABLE LABORATORY EQUIPMENT

TRAINING AIDS:

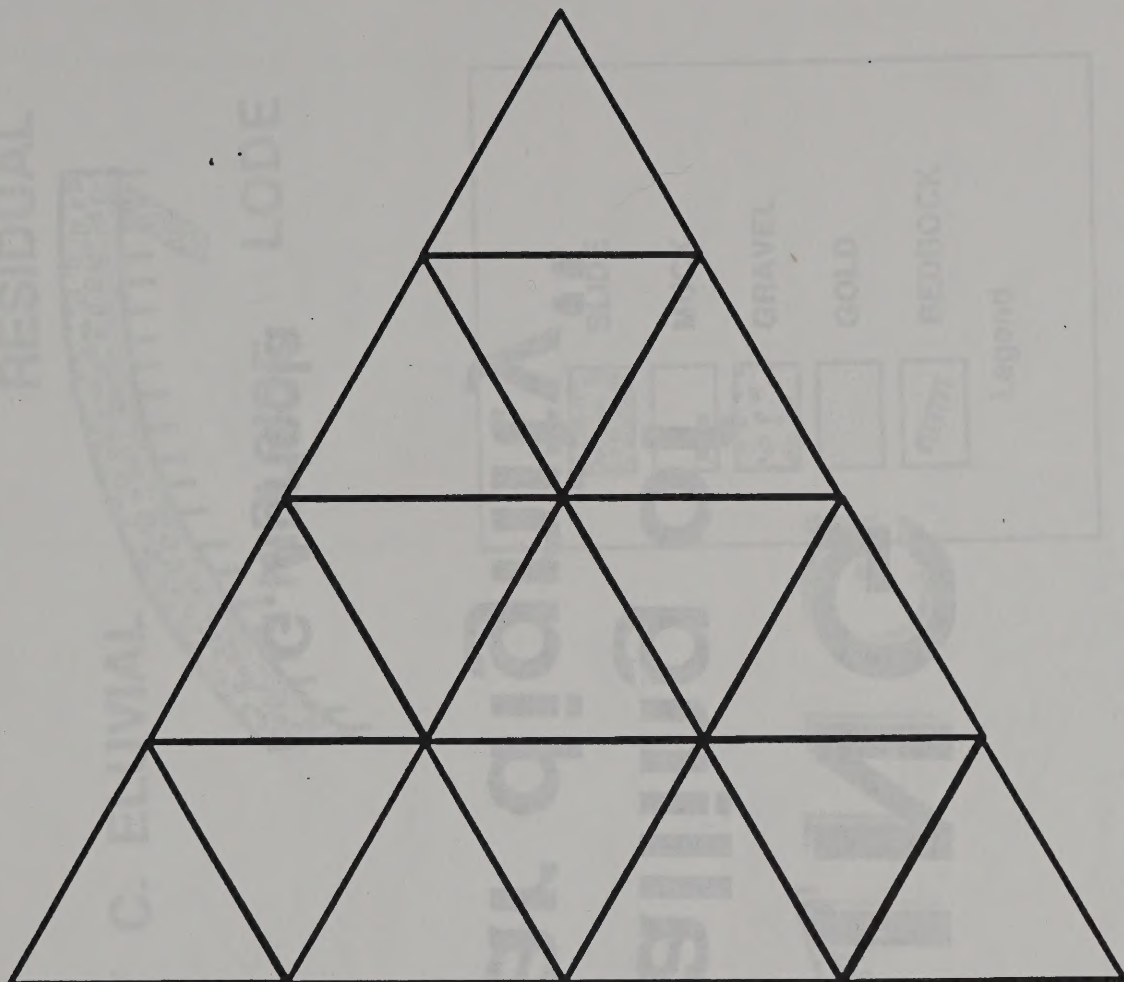
EQUIPMENT, MATERIALS, REFERENCED:

TIME	SUBJECT MATTER	METHOD	INSTRUCTOR ACTIVITIES
	Describe general lab setting		OH 16
	Describe lab procedures for placer		Show Video
20 mins	Lab Procedures - Placer		OH 16
	1. Dry sample in drying oven or on an electric heating element.		
	2. Separate magnetite from concentrates with magnet.		
	3. Separate free gold from concentrates using:		
	a. Heavy liquid medium		
	b. Wilfrey Table		
	c. Repanning		OH 17
	d. Gold Hound		
	4. Dry gold		
	5. Screen gold using 10, 20, 40, 100 mesh screens.		
	6. Microscope analysis		
	7. Weight gold remaining on each screen		
	8. If black sand still present with gold		
	9. Place concentrates from no.7 in glass mill jar		
	10. Add a globule of clean, gold free mercury the size of a bean to the jar.		
	11. Place jar on rollers and roll for 1/2 to 1 hour.		
	12. Remove amalgam and place in 250 ml beaker.		
	13. Add 40-50 ml dilute (1:1) nitric acid and place on electric heating element.		
	14. Digest until mercury is match-head in size.		
	15. Transfer to No. 0 parting cup.		
	16. Add fresh dilute acid.		
	17. Digest as required over low heat to a gold sponge.		
	18. Decant off any remaining gold.		

19. Wash sponge 3-5 times with warm distilled water.
20. Add 2-3 drops of alcohol.
21. Dry parting cup over low heat.
22. Anneal gold in cup by bringing bottom of cup to dull red heat in a furnace or on a Bunsen burner.
23. Transfer gold to Metler balance.
24. Weight sample to nearest .1 mg.
25. Fill out Placer Summary Sheet and field notebook.

END OF LESSON NINE.

Keill's
Overheads

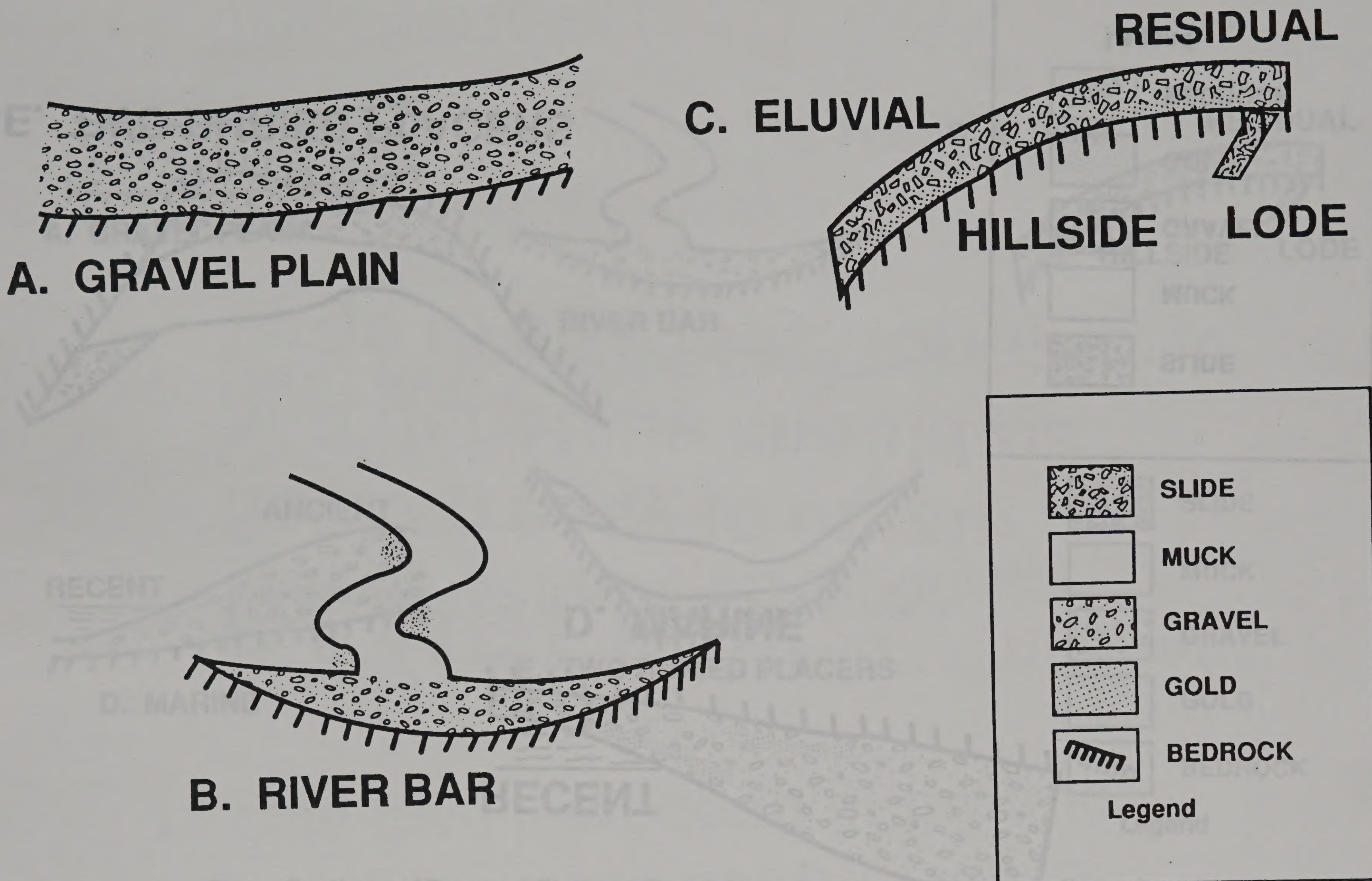


How many triangles?

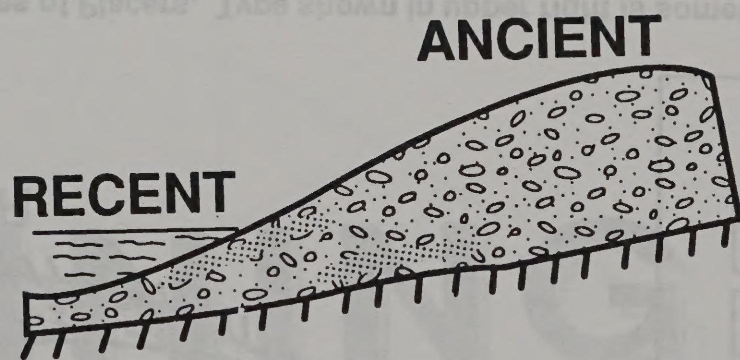
MINING

"... a calling of peculiar dignity"

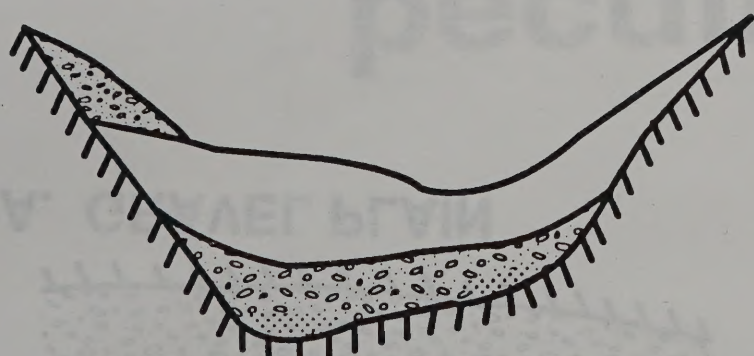
G. Agricola



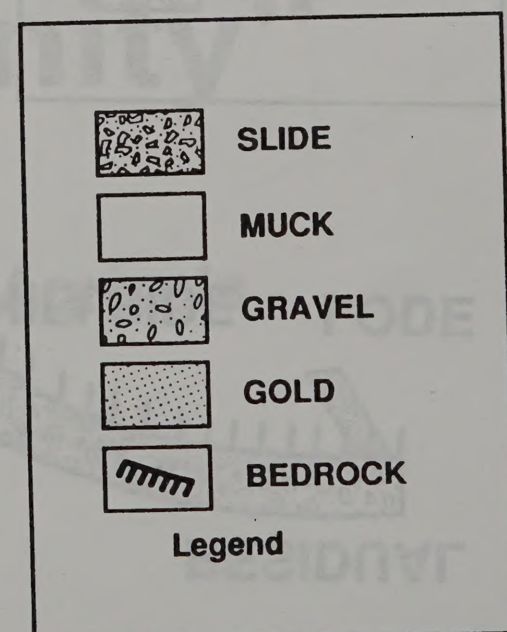
Cross Sections of Types of Placers. Type shown in upper right is sometimes found in glaciated areas.



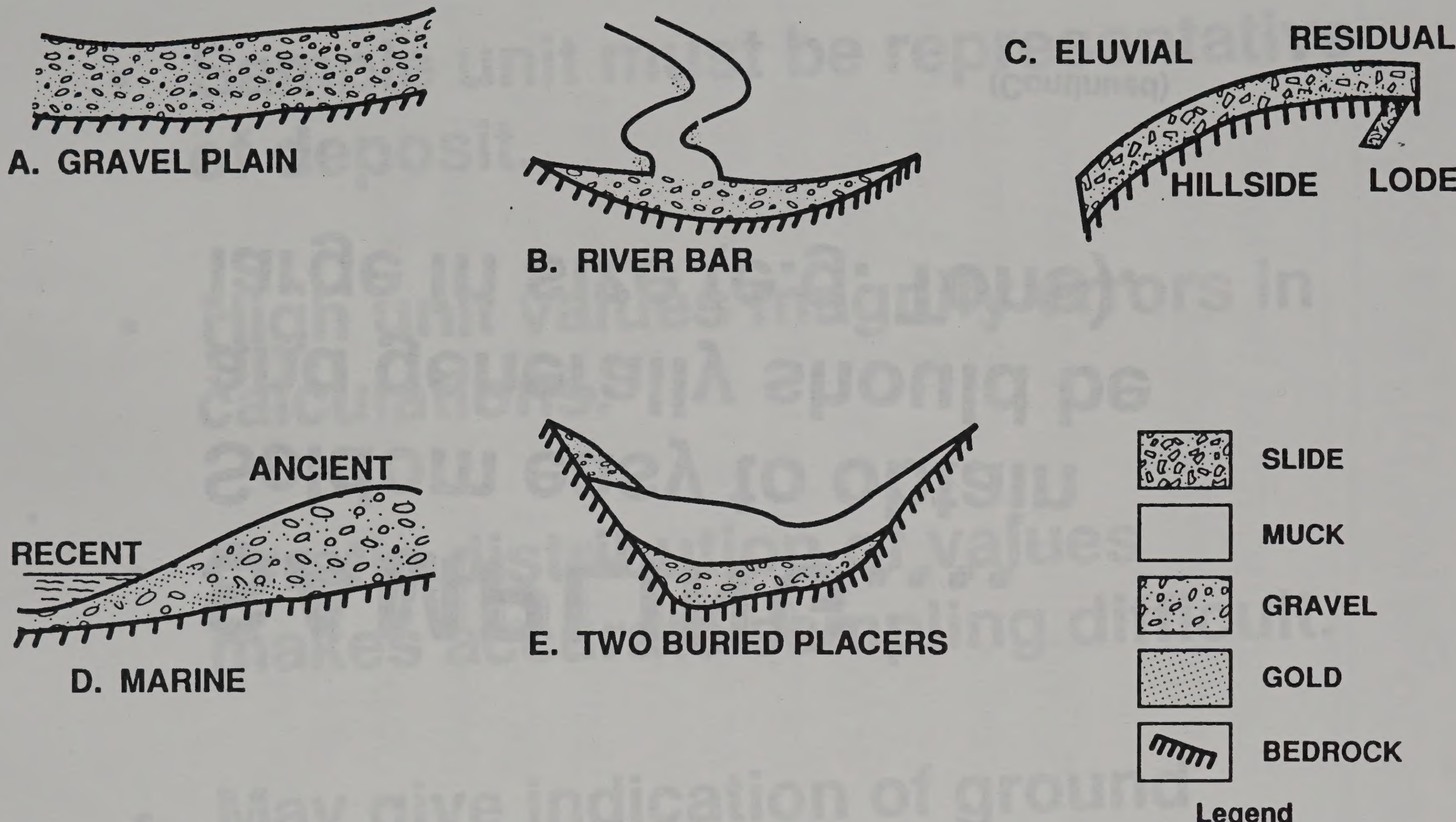
D. MARINE



E. TWO BURIED PLACERS



Cross Sections of Types of Placers.



Cross Sections of Types of Placers. Type shown in upper right is sometimes found in glaciated areas.

SAMPLING....

Seldom easy to obtain and generally should be large in size (e.g. Tons).

(Continued)

SAMPLING (Cont.)

- Sample unit must be representative of deposit.
- High unit values magnify errors in calculations.
- Erratic distribution of values makes accurate sampling difficult.
- May give indication of ground conditions but not true value.

Rules of Sampling

1. Sampling must be representative of the whole lot of material.

2. Sampling , at each parting or reduction in amount, must still represent the whole lot and any errors or inconsistencies introduced must be controllable or accounted for mathematically.

3. Sampling accuracy relates to the amount of control one has over the sampling correctness and is dependent upon the characteristics of control.

Sample Selecting Process

Probabilistic

- well adapted to 3 dimensional deposits
- all elements submitted for selection
- have given probability for being selected
- works well on compact ore bodies
- well studied by geostaticians

Sample Selecting Process

Non -Probabilistic

- selection not founded on notion of probability
- typically used on particulate materials
- deterministic
- purposive
- difficult to select increment and extract representative sample

Non-Probabilistic Sampling

- no theoretical approach to selecting samples
- sampling errors large enough to deprive sample of any practical value (specimen)
- used by many commercial firms, in sampling, mine evaluation, pilot plants, evaluation and for process control

Common Properties of Non-Probabilistic Sampling Processes

1. An important fraction of the lot is submitted for sampling with zero chance of being selected

2. Not possible to logically connect the various sampling errors to the mode of selection

3. The sampling is always biased to some extent

Conclusions: no sample generated by a non-probabilistic selecting process should be used to reach important financial decisions! (eg. validity, mine development, product purchase, compliance with regulations)

Purposive Sampling: Chip or Shovel

- if part of the lot not available, operator selects best way from the most accessible areas

- accuracy of sampling operation depends on operator's choice

- operator not uniform or consistent

- operator may have biased concept of what the samples are to prove

- sampling therefore is not accurate and very likely inequitable in representing the lot

Deterministic Sampling and Grab, Auger, Drill

- operator collects increments from the most accessible part of the lot

- due to the segregation of particles, operator tries to collect as many increments as practical

- large portion of the lot is systematically not available to the sample tool or procedure
  - bottom of truck or rail car
  - bottom of drum
  - center of pile or load bucket

(Continued)

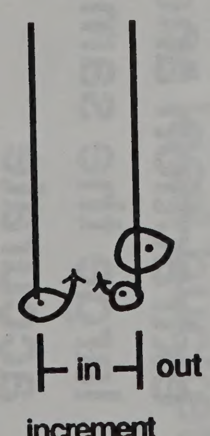
Deterministic Sampling and Grab, Auger, Drill (Cont)

- drills, augers are only improved grab sample
- these are probabilistic only when the sample point can be selected by probability (rare)
- sample points usually pre-selected which is non-probabilistic
- if sample points were probabilistic, errors from increment delimiting, extraction and heterogeneity are so large the sample results are not accurate

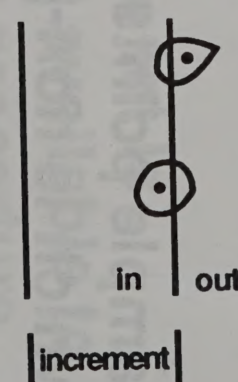
Center of Gravity Rule

All fragments having their center of gravity inside the selected increment boundaries belong to the increment.

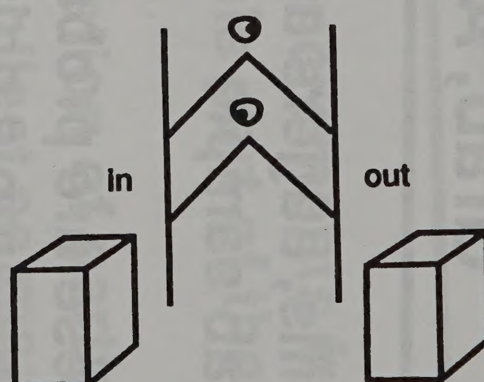
Drill



Channel



Spliter



ADEQUATE SIZE

Largest Rock	Sample Size
+ 10 in.	3 tons
+ 4 in.	1 ton
+ 2 in.	400 lb.
+ 1 in.	100 lb.
- 1/2 in.	50 lb.

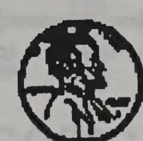
EFFECT OF PARTICLE OF Au IN SAMPLE (adapted from J. Wells)

SAMPLE	PARTICLE SIZE @ $400 per oz. 20 mesh (6.5 mg)	40 (1 mg)	60 (.3 mg)
7 1/2" X 12" Drill Hole	$6.50/yd	$100/yd	$.33/yd
5 1/2" X 12" Drill Hole	$14.82/yd	$2.28/yd	$.68/yd
3" X 12" Drill Hole	$41.34/yd	$6.36/yd	$1.91/yd
6" X 12" Channel	$4.64/yd	$.63/yd	$.24/yd
12" X 12" Channel	$2.31/yd	$.33/yd	$.09/yd
16" PAN@180 PAN/yd	$15.39/yd	$2.13/yd	$.63/yd

Gold Particle Size Estimation

	Coarse	Med	Fine	V. Fine	Flour
MESH (USS)	+10	-10+20	-20+40	-40+60	-60
AVG. WT.(mg)	-	6.0	1.56	0.31	0.05
AVG. NO./g	-	170	340	3,200	20,000

-20+40
mesh
1.0 mg

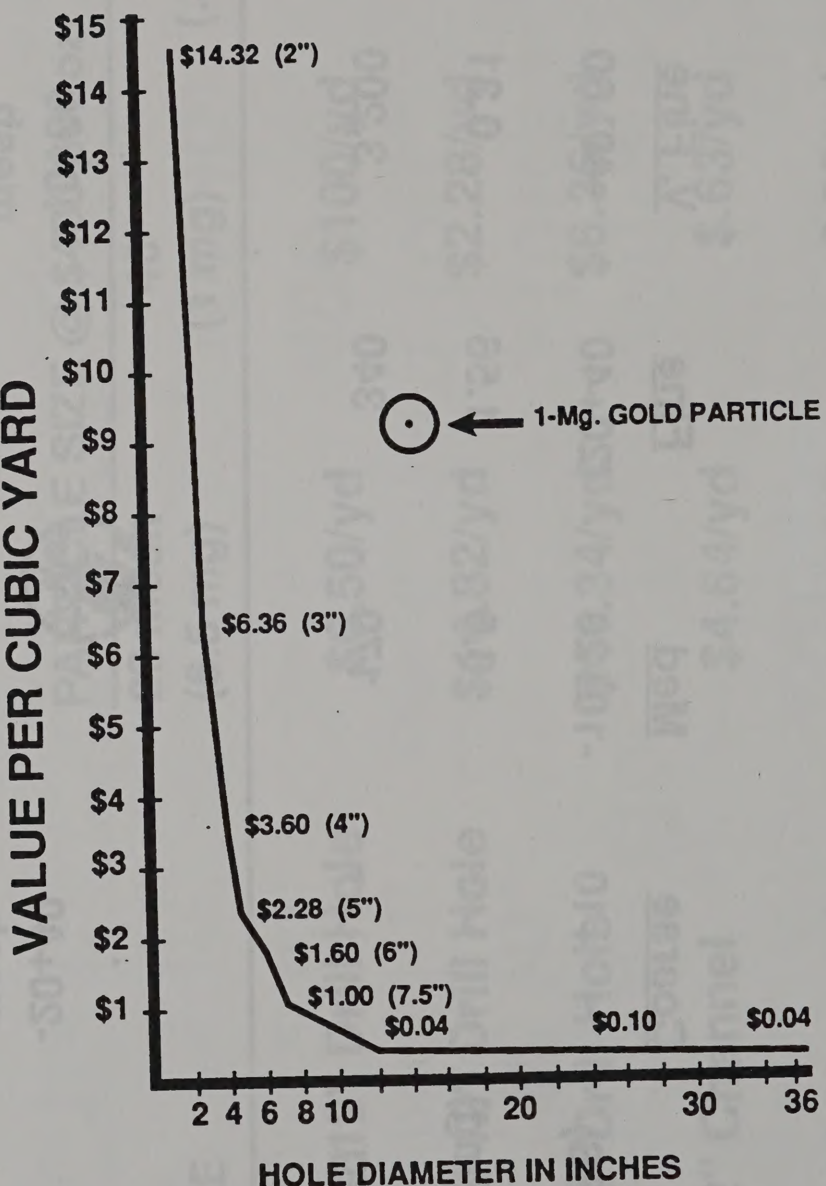


1 cent

-40+60
mesh
.5 mg

Effect of a 1-Mg. Particle of Placer Gold on Calculated Value of a 1-Foot Sample Increment*

*Gold at $400 per ounce



Curve showing the calculated value which would result from a single 1-Mg. particle of 900-fine placer gold (at $400/oz). It can be seen that small-diameter drill holes are extremely sensitive to stray gold particles.

HO-3K9-DK-21

Controllable Sample Factors or Errors

- increment to be sampled
- extraction of increment "Center of Gravity Rule"
- contamination of sample
- chemical or physical alteration of sample
- preparation of errors
- analytical errors
- human errors, intentional or unintentional

SAMPLING ERRORS AND THE RELIABILITY OF SAMPLING*

Errors common to all placer environments

- Incorrect sample dimensions
- Faulty analytical procedures
- Salting
- Carelessness
- Inaccurate measurements
- Drilling into a hard bottom
- Use of drilling correction factors
- Inaccurate logging
- Unsuitable drilling equipment
- Uneven distribution of values
- Not allowing for dilution and batters
- Ignorance

(Continued)

SAMPLING ERRORS AND THE RELIABILITY OF SAMPLING*
(Cont.)

Errors in continental placer sampling

- Splitting samples
- Irregular-shaped pits and channels

Errors in transitional placer sampling

- Grain counting
- Failure to recognize changes in lithology

Errors in marine placer sampling

*Source: Aluvial Mining, by Eoin MacDonald, 1983, pages 227-233

Preparation Error

Errors resulting from contamination
- by dust
- by material present in sampling circuit
- by abrasion
- by corrosion

Errors resulting from losses
- as dust
- as material left in sampling circuit
- of a specific fraction of the sample

Errors resulting from alteration of the composition

Errors resulting from alteration of the physical composition

Errors resulting from unintentional mistakes

Errors resulting from fraud or sabotage

Errors Resulting from Unintentional Mistakes

For psychological, political, or economic reasons, sampling within the estimation circuit is the most neglected.

This is true for most countries to the point that it is not even taught at universities or colleges.

As a result the sampling operator is unaware of the most elementary notions of sampling correctness.

He blindly follows a vague recipe based on empirical observations, necessarily introducing unintentional mistakes.

- Dropping of samples followed by incomplete recovery
- Mixing of fractions belonging to different samples
- Mixing labels
- Poor maintenance of sampling equipment
- Contamination and losses

SAMPLING CONSIDERATIONS

1. Adequacy of size to obtain representive sample.

2. Uniform width and depth of channel from top to bottom.

3. Adequacy of sample to character of the ground.

4. Spillage of sample while collecting.

(Continued)

SAMPLING CONSIDERATIONS
(Cont.)

5. Spillage of sample while processing.

6. Water washing bedrock just prior to collecting sample.

7. Erratic collection of samples from backhoe bucket - should sample every bucket.

8. Drill mast not vertical.

9. Natural and introduced oils in the sample: Avoid !!

(Continued)

SAMPLING CONSIDERATIONS
(Cont.)

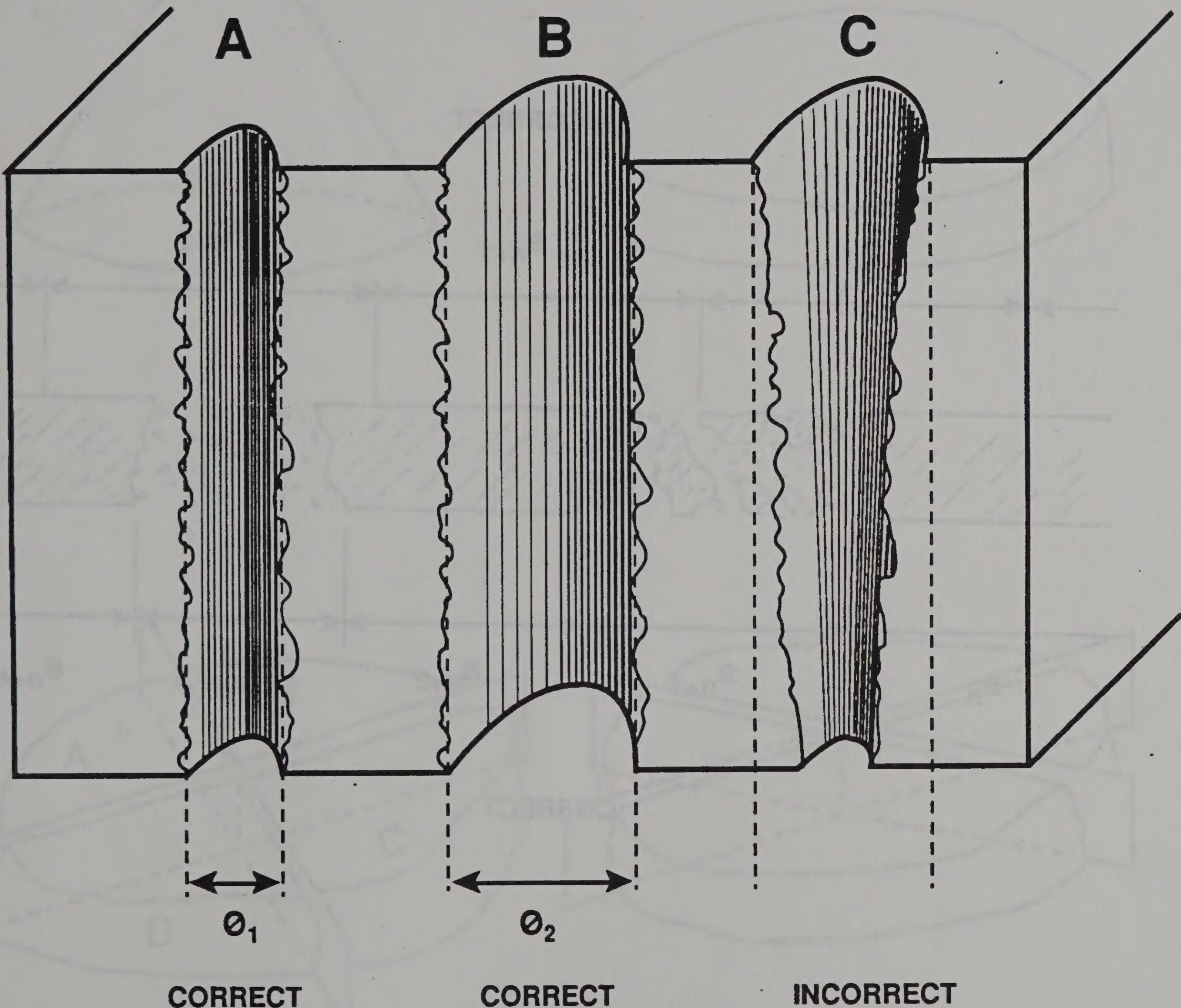
10. Soil running into drill hole: dilution.

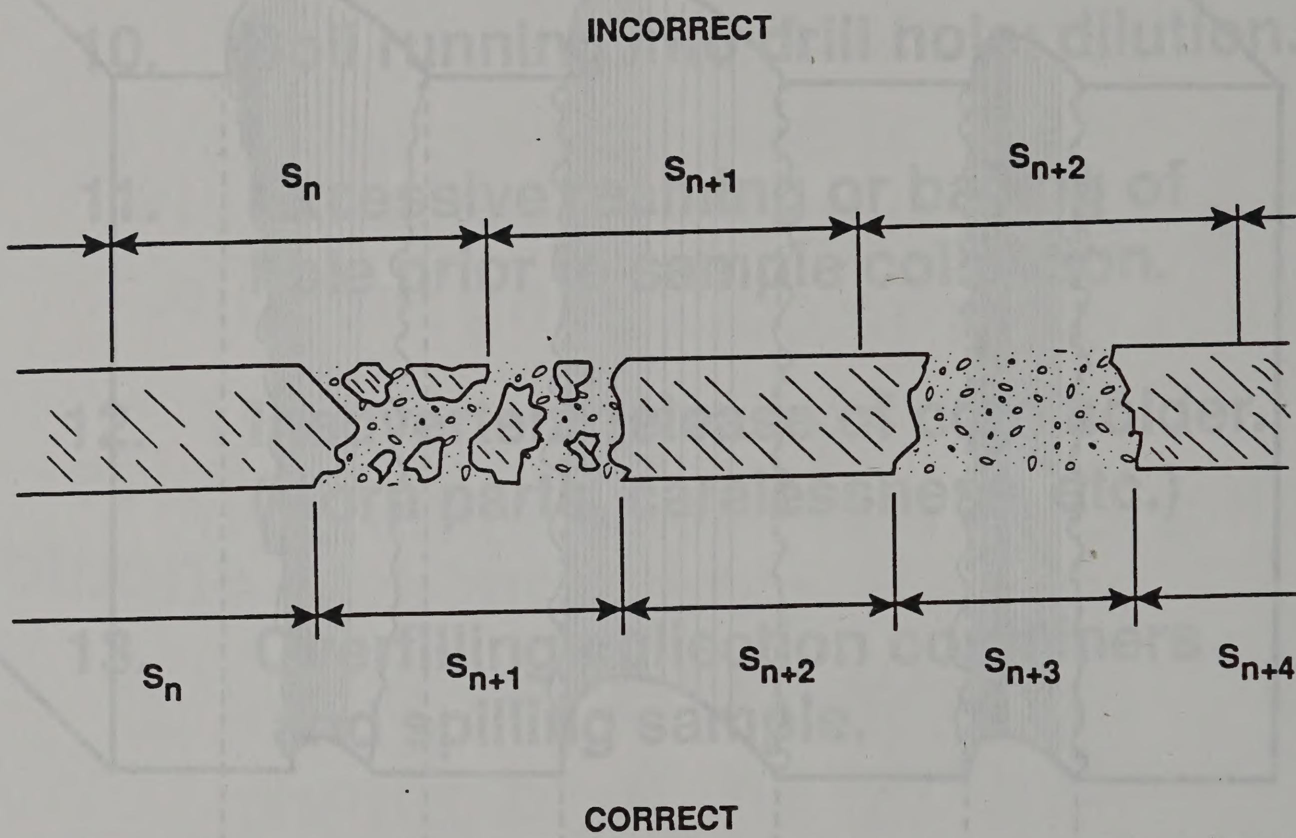
11. Excessive reaming or bailing of hole prior to sample collection.

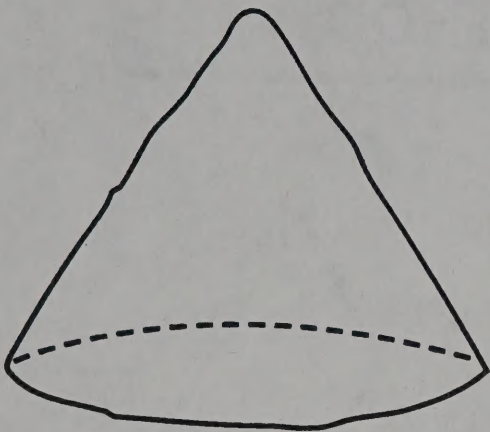
12. Inadvertent release of core holder. (worn parts, carelessness, etc.)

13. Overfilling collection containers and spilling sample.

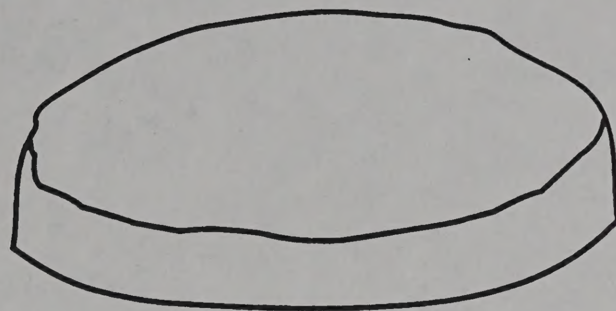
14. Dirty equipment of both applicant and BLM. Clean all tools and machines prior to running each sample.


A
B
C
$Ø_1$
$Ø_2$
CORRECT
CORRECT
INCORRECT

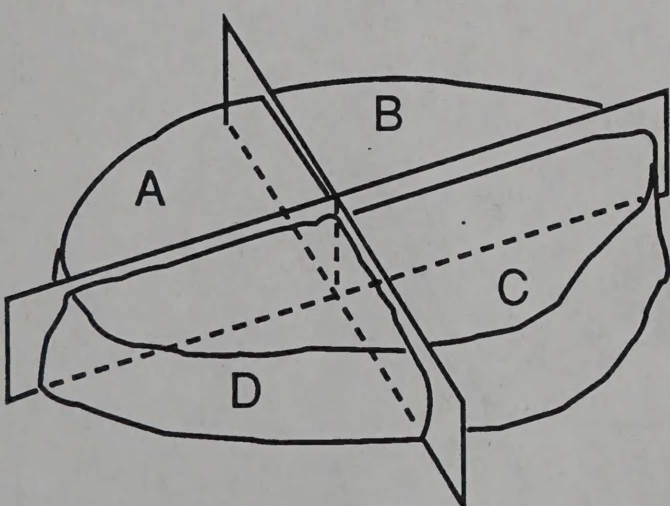

INCORRECT
S_n
S_{n+1}
S_{n+2}
S_n
S_{n+1}
S_{n+2}
S_{n+3}
S_{n+4}
CORRECT



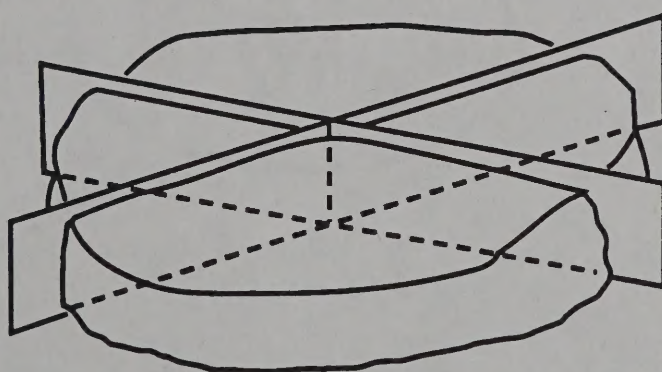
A



B



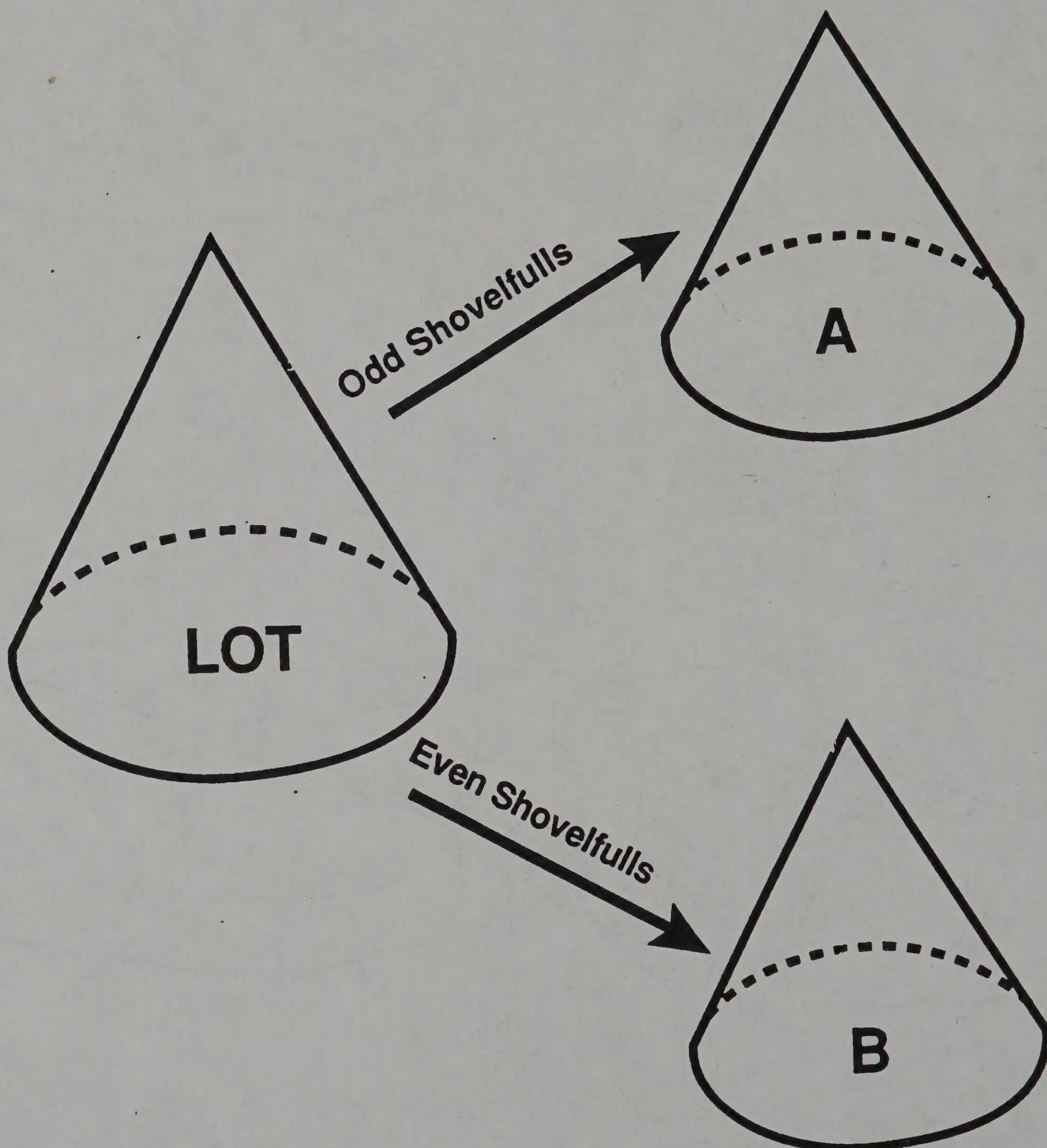
C

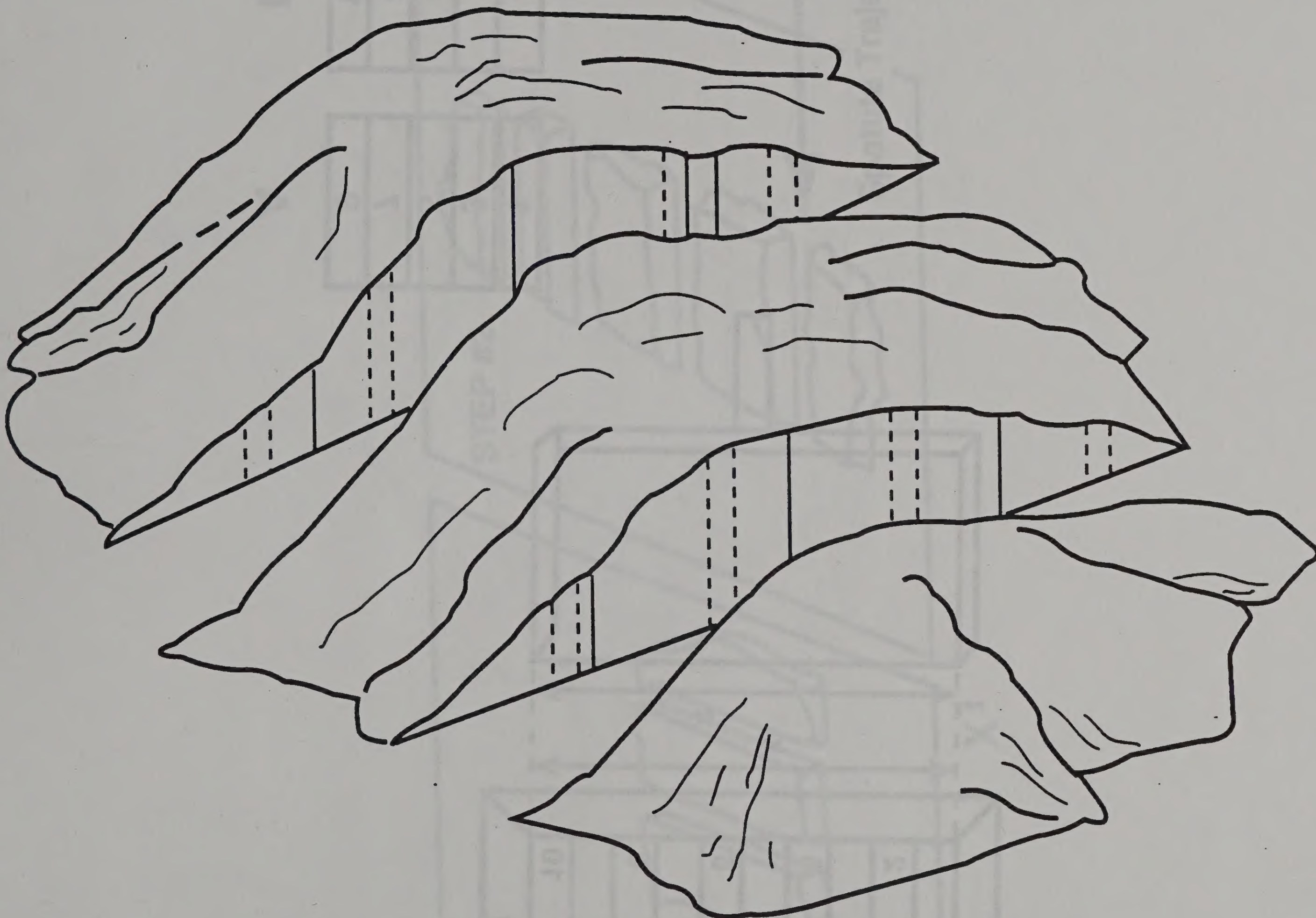


D

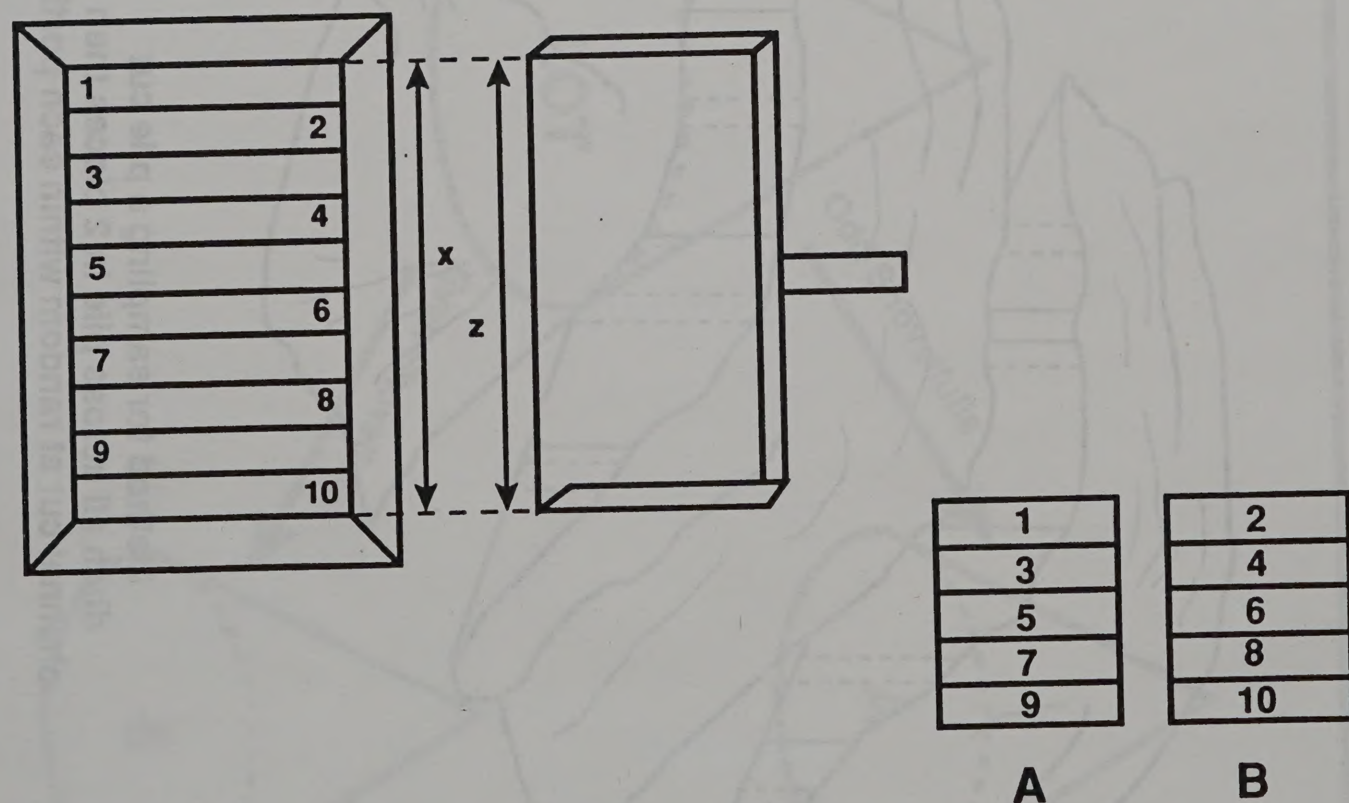
Incorrect method
biased samples

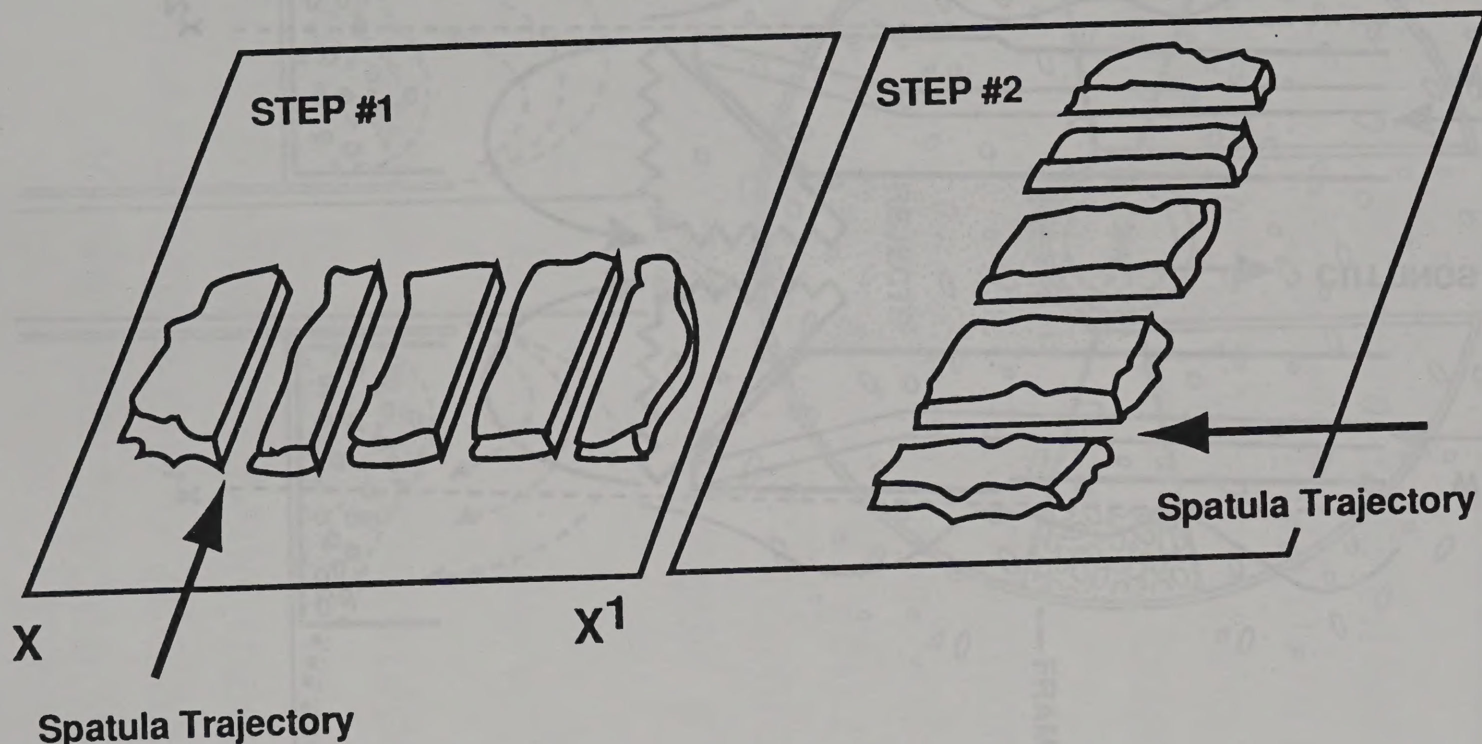
CORRECT METHOD



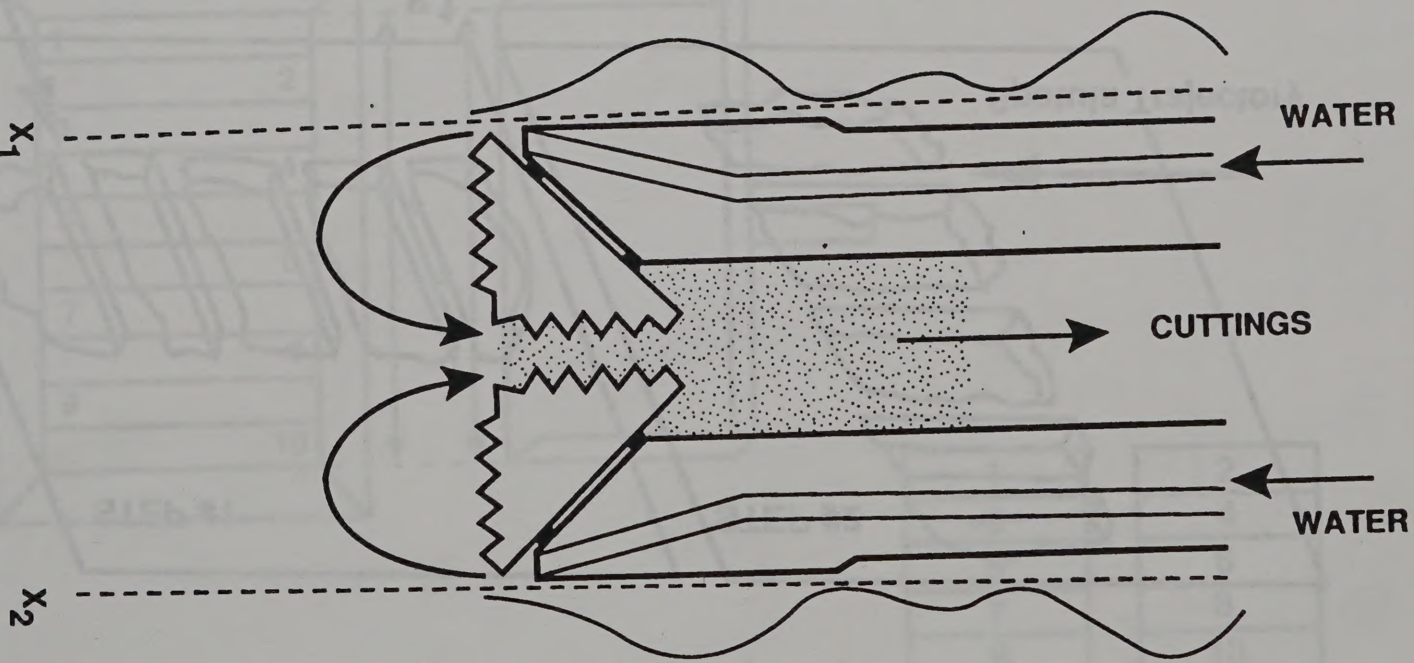


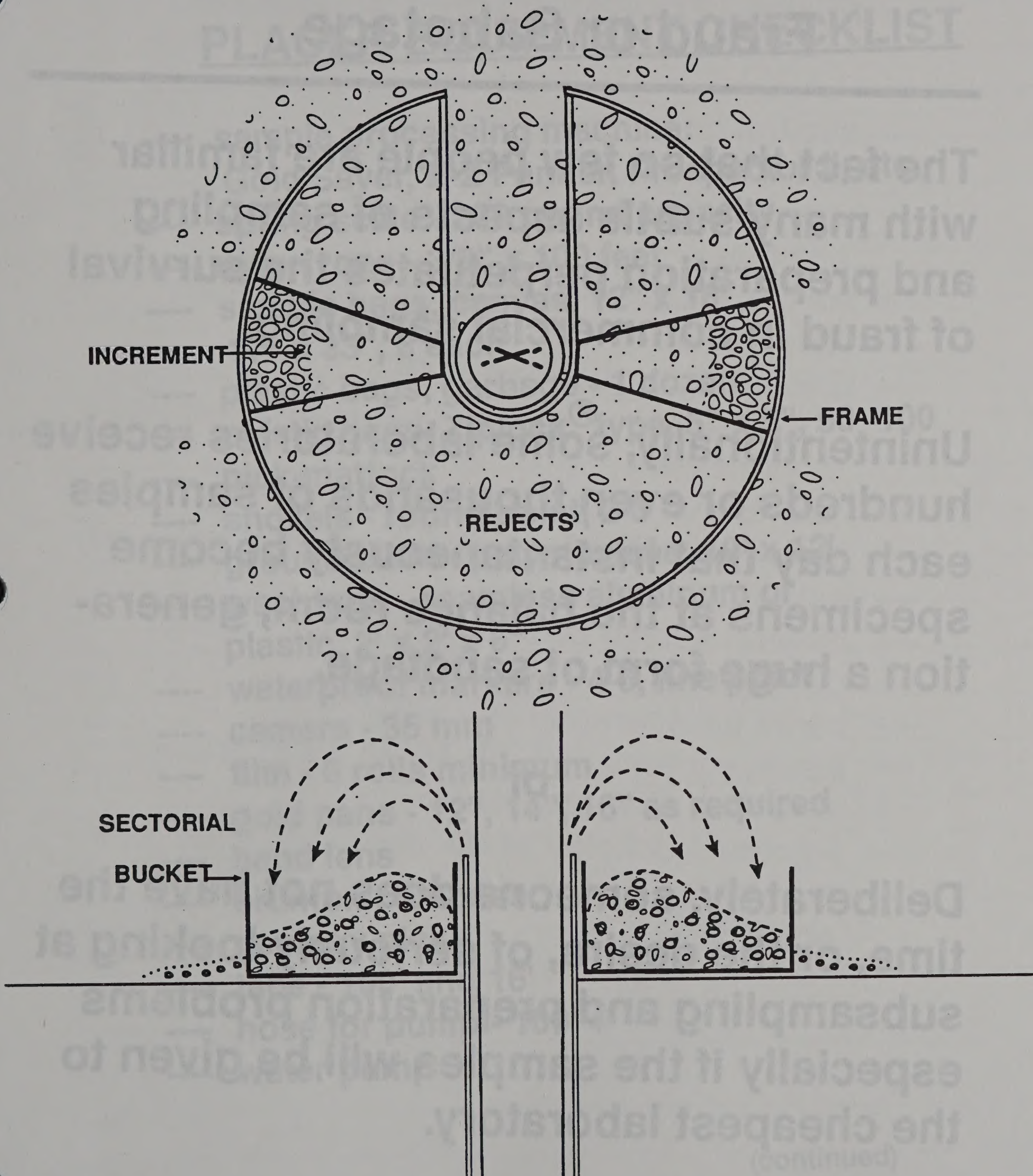
Method for sampling a pile cut ditch all across pile in 2 places then channel cut at random within each portion


1
2
3
4
5
6
7
8
9
10
x
z
1
3
5
7
9
2
4
6
8
10
A
B



HO-3K9-DK-36


x_1
WATER
CUTTINGS
WATER
x_2


INCREMENT
FRAME
REJECTS
SECTORIAL
BUCKET

Errors Resulting from Fraud or Sabotage

The fact that so few people are familiar with many subtle aspects of sampling and preparation perpetuates the survival of fraud in commercial sampling.

Unintentionally, some laboratories receive hundreds or even thousands of samples each day that instantaneously become specimens at the balance room, genera-tion a huge form of sabotage.

or

Deliberately, someone does not have the time, or the desire, of seriously looking at subsampling and preparation problems especially if the samples will be given to the cheapest laboratory.

PLACER EQUIPMENT CHECKLIST

---- sample processing machine:
  Gold Saver, E-Z Panner, Prospector II, etc.
---- spring scale - 200 pound capacity
---- nylon rope - 3/8" x 100 feet
---- sample bags, canvas, 12" x 18",
  17" x 33", 6 each
---- plastic bags, garbage - 1 dozen
---- plastic bags, Ziplock® type, 5" x 5", 50 -100
---- pick-mattock
---- shovels - round point, 2-3
---- ground cloth - polyethylene, 10' x 12'
---- washtubs - seamless aluminum or
  plastic, 2' x 2' x 6"
---- waterproof markers - 4-6, fine point
---- camera - 35 mm
---- film - 6 rolls minimum
---- gold pans - 12", 14", 16" as required
---- hand lens
---- brunton compass
---- geology pick
---- tape - 100' and 16'
---- hose for pump - 100' +
---- water pump

(continued)

PLACER EQUIPMENT CHECKLIST (Cont.)

---- suction hose for water pump
---- tool kit
---- spare parts for engines and processing machines
---- survey flagging - fluorescent pink, yellow
---- vials - 20 dram
---- dispersing agent - Calgon®
---- 1" x 2" lath or waterproof paper for sample identification
---- field forms: 3842-1, 3842-2
---- placer checklist form - (10-page form optional)
---- topographic maps
---- mineral survey plat - if patent examination
---- geologic map - if available
---- aerial photography - if available
---- sizing screens - 10, 20, 40, 60 mesh, 3-inch or 8-inch size, stackable
---- 5-gallon buckets - 15-20, plastic with handles
---- generator - if needed to power processing equipment

Form 3842–1
(December 1977)

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF LAND MANAGEMENT

PLACER SAMPLE FIELD RECORD

Sample number*

Serial number

Date

State | County

Claim

Give legal description: Township | Range | Section | Meridian

Type of deposit | Type of sample | Dimensions of cut

Place taken**

INTERVAL		FORMATION
FROM	TO	

Bedrock ☐ Yes ☐ No | Type

Overburden: Feet | Type | Water level ft.

Sample weight | Sample volume ☐ Loose measure ☐ Bank measure

Boulders ☐ Yes ☐ No | Average size | Maximum size | Estimated percent of bank-run

Cement ☐ Yes ☐ No | Remarks

Clay ☐ Yes ☐ No | Remarks
Estimated percent

Caliche ☐ Yes ☐ No | Remarks
Estimated percent

Hard digging ☐ Yes ☐ No | Remarks

Sampled by

Others present

Time: Arrive | Depart | Weather

* *Carry this number forward to Form 3842–2* (Placer Sample Processing Record)
** *Place sketch or remarks on reverse*

(Continued on reverse)

HO-3K9-DK-42

DIAGRAM

Remarks

Form 3842–2
(December 1977)

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF LAND MANAGEMENT

PLACER SAMPLE PROCESSING RECORD

Sample number *	
Serial number	
Date	

1a. Dry weight *(lbs)*	b. Volume	c. How measured	d. How processed

2a. Visible gold *(colors)* ** ☐ No. 3 ☐ No. 2 ☐ No. 1 ☐ None visible

b. Weight *(milligrams)*

c. Estimated fineness

3a. Gold removed ☐ Manually ☐ By amalgamation ☐ Other *(specify)*

b. Does gold amalgamate readily? ☐ Yes ☐ No *(if "no," explain)*

4a. Character of Gold ☐ fine or flaky ☐ coarse or shotty ☐ smooth ☐ rough

b. Give weight of black sand in sample , and pounds black sand per cubic yard

c. Was screen used in washing process? ☐ Yes ☐ No *(if "no," explain)*

d. SIZE OF OPENINGS

Undersize (%)	Oversize (%)

e. Material washed ☐ easy ☐ normal ☐ difficult *(explain)*

f. Grade of sluice or rocker *(inches per foot)*

Remarks

Processed by

Others present

*Number must be same as Form 3842–1 (Placer Sampler Field Record)
***#3 colors consist of gold particles weighing less than 1 milligram;*
#2 colors weigh between 1 and 4 milligrams;
#1 colors weigh over 4 milligrams.
Weigh and note individual colors weighing 10 milligrams or more.

GPO 843–580

HO-3K9-DK-43

THIS PAGE
INTENTIONALLY
BLANK

Form 3842–3
(December 1977)

UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF LAND MANAGEMENT

Sheet of

Serial number

Date

PLACER SAMPLE SUMMARY SHEET

State	County	Legal description

SAMPLE NUMBER *	WHERE TAKEN	NET WEIGHT OR VOLUME	HOW PROCESSED	RECOVERED GOLD (mg)	CENTS Au @ $ /oz	CENTS PER CUBIC YD	NOTES

Sampled by	Processed by	Calculated by

Remarks

HO-3K9-DK-44

THIS PAGE
INTENTIONALLY
BLANK

Percent of Recovery by Type of Equipment

Hand Shovel	100	unless large run
Suction Dredge	70-80	
Back Hoe Buckets	90-100	
Auger	20-50	
Banker Drill	0-100	unknown
Hand Churn Drill	90-100	needs correction factors
Light Churn Drill	90-100	"
Heavy Churn Drill	90-100	"
Hand dug Cassions	80-100	
Machine Dug Cassions	80-100	
Drill-Percussion	80-95	needs correction factors
Drill-Rotary	40-75	"
Drill-Reverse Circ.	90-100	"
Drill-Vibratory	80-90	"
Drill-Resonant	95-100	"

EFFICIENCIES OF SAMPLING DEVICES

GOLD PAN	nuggets to 100 microns	99 - 100%
SLUICE BOX		
Hungarian riffles	nuggets to 200 mesh	70 - 80%
Chinese riffles	nuggets to 200 mesh	80 - 90%
regular riffles	nuggets to 100 mesh	50 - 90%
ROCKER BOX	nuggets to 200 mesh	75 - 90%

(Continued)

EFFICIENCIES OF SAMPLING DEVICES (Cont.)

GOLD SAVER (trommel)	- 1/4" to 100 mesh	85 - 92%
E-Z PANNER (vibrating screen)	- 3/4" to 200 mesh	85 - 96%
AIR BELLOWS (dry washers)	- 1/4" to 100 mesh	70 - 85%
SPIRAL WHEEL (goldhound)	- 20 to 250 mesh	80 - 90%

TO INCREASE EFFICIENCY. . .

- Sluice Box: Increase grade from $1^1/_2$":1' to $2^1/_4$":1'
- Increase water percent of feed slurry (i.e. decrease density of feed)
- Add a wetting agent: soap or shampoo
- Add an acid (dilute) to feed water
- Classify feed by screening

Recommendations for Sluice Box Recovery

Rany Clarkson
Klondike Miners Association, 1989

- Screen feed to - 3/4".
- Set box at 2.5' to 3" per foot gradient.
- Use single channel box. 30 foot minimum length.
- Use alternating angle iron & expanded metal screen (5/8") riffle sections on a 1:2 length ratio with 2'slick plate at front of first riffles.
- Set angle iron riffles at -15 upstream slope and 1" high, maximum.
- Set gap between riffles at 2".
- Feed box at maximum of 8 yd 3/ft of width.
- Set water usage at 192 gpm/ft of width in expanded metal section, 432 gpm/ft of width in riffle section.

Conclusions from University of Toronto, British Columbia Placer Mining Study, 1987-1988

- 3 channel sluice box not as efficient as single channel with screen classification due to loss of turbulence and non-uniform feed rates to the side boxes.

- Best performance in sluices obtained by:
  - Classification to minus 3/4 inch feed to riffles
  - Screen feed well, concentrate with either Hungarian riffles over expanded metal or alternate sections of each

(Continued)

Placer Mining Study, 1987-1988 (Cont.)

- 100% recovery with riffles: Hungarian: + 28 mesh gold
  Expanded metal: - 48 mesh gold
- Feed at 6 yd^3 /hour/foot of width of box
- Keep water to 200 to 400 gpm/yd^3 processed
- Keep recycle water at 20% solid slurry
- Keep clay concentration below 60 grams/liter, optimum recovery rates in riffles and minimum wear on pump found at 10g/1
- Flocculents good only in low flow of water, less than 500 gpm, good for final treatment of discharge from last pond

RECOVERY RATES*

PROCESSING	GOLD RECOVERY	
	+10 mesh	+60 mesh
· Slick Plate	70%	20%
· Grizzly 6" openings	90%	80%
· Dbl Screen, vibrating	90%	80%
· Conveyor feeding dbl screen	90%	90%
· Trommel screen	80 - 90%	

* from Univ. Alaska, Mineral Industry Research Lab. 1987-1988 studies

Computing Placer Sample Values[*1]

1. Based on Volume Measurement:

$$\text{¢ Per Cubic Yard} = \frac{\text{Milligrams Recovered} \times \text{Price Per Mg (¢)} \times 27}{\text{Volume of Sample (cu. ft.)}}$$

*1 Indicated Value, No Fineness Correction

Computing Placer Sample Values*2 HO-

3K9-DK-26HO-3K9-DK-26

2. Based on Weight Measurement:

$$¢\ \text{Per Cubic Yard} = \frac{\text{Milligrams Recovered} \times \text{Price Per Mg } (¢) \times 3{,}000 \quad {}^{*2}}{\text{Weight of Sample (pounds)}}$$

*2 Where One Bank-Yard = 3,000 Pounds.

Computing Placer Sample Values*3

3. Based on Pan Measurements:

$$\text{¢ Per Cubic Yard} = \frac{\text{Milligrams Recovered x Price Per Mg (¢) x Pan Factor *3}}{\text{Number of Pans in Sample}}$$

*3 Number of Pans Equivalent to One Cubic Yard in Place

180 pans - 16" diameter

400 pans - 12" diameter

Value Calculations

$$\text{¢/mg} = \frac{\text{Price of Au in cents/oz} \times \text{Fineness}}{\text{31,104 mg/oz}}$$

$$\text{\$/yd}^3 = \frac{\text{Recovery Au in mg} \times \text{value in ¢/mg} \times 27\ \text{ft}^3\ /\text{yd}^3}{(\text{Size of Sample in ft}^3) \times 100\ \text{¢/\$} \times \text{concentrate recovery \%}}$$

Size of Sample if not in Cubic Feet: from Rock Density Tables or Weight Tables Multiply: lb/cu. ft. x 27 ft^3 = lb/yd^3

Examples:

Quartzite Bedrock: from table = 114 lb/ft^3

lb/yd = 114 x 27 = 3,078 lb/yd^3

$$\text{¢/mg} = \frac{30{,}000\ \text{¢/oz} \times .90}{31{,}104\ \text{mg/oz}} = .86\ \text{¢/mg}$$

Bulk Sample

$$\text{\$/yd}^3 = \frac{50\ \text{mg} \times .86\text{¢/mg}}{\dfrac{484\ \text{lb}}{3{,}078\ \text{lb/yd}^3} \times 100} = \frac{43}{15.72} = \text{\$2.74/yd}^3$$

Channel Sample

$$\text{\$/yd}^3 = \frac{50\ \text{mg} \times .86\text{¢/mg} \times 27\ \text{ft}^3\text{/yd}^3}{(1' \times .66' \times 5') \times 100\ \text{¢/\$}} = \frac{1161}{330} = \text{\$3.52/yd}^3$$

Value Calculations (cont.)

Drill Hole

$$\$/YD^3 = \frac{mg\ (Au) \times ¢/mg \times 27\ ft^3/yd^3}{(\text{Sample size in } ft^3) \times (CR) \times (DR) \times 100\ ¢/\$}$$

CR = % Concentrator Recovery of Sample Device

DR = % Recovery of the Drill

Example

6" churn drill hole: .18 ft^3/foot of sample hole

10' pay gravel

CR = 90%, DR = 95%

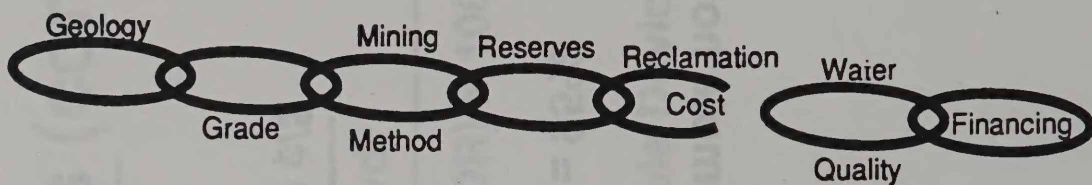
$$\$YD^3 = \frac{50\text{ mg} \times .86¢/\text{mg} \times 27\ ft^3/yd^3}{(.18\ ft^3/ft \times 10ft) \times .90 \times .95 \times 100\ ¢/\$}$$

$$= \$7.54/yd^3$$

How Thorough Should a Mineral Investigation Be?

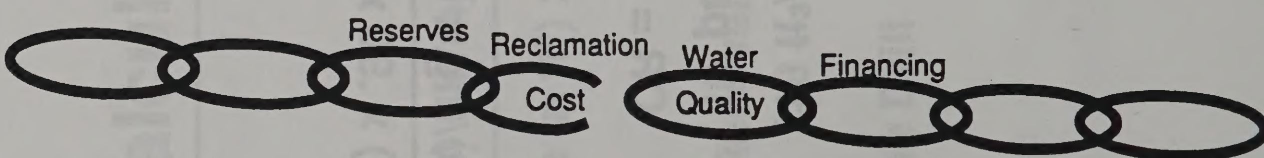
Private Industry

The examiner tests successive links in a chain of facts: When one link is found weak- - the examination is over.

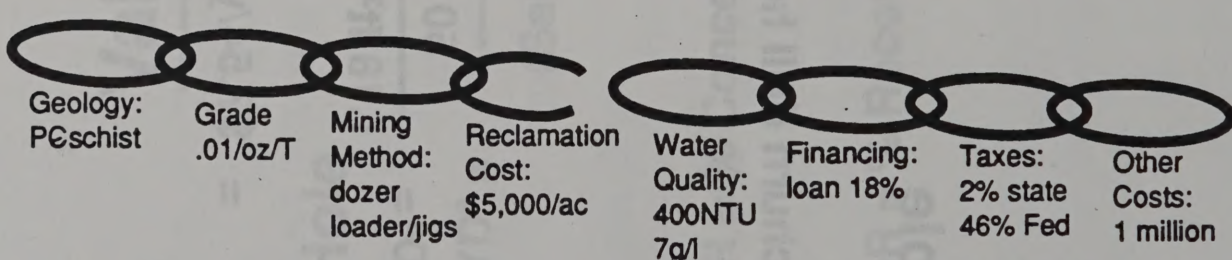


Government

We must go beyond pointing out the fatal link.



We must show detailed knowledge of the property.



How Thorough?

Whatever is necessary to support the conclusions of the examiner.

THE ABCs OF REPORT WRITING

On the first page of your report:

A. Identify the problem or question involved.

B. Mention or discuss briefly all pertinent factors.

C. Recommend the proper action to be taken, or not to be taken.

The three Cs of a good report:

It is - Clear

Concise

Complete

(Continued)

THE ABCs OF REPORT WRITING (Cont.)

- Reports are usually too long rather than too short.

- Mineral reports will be read by laymen for decision-making purposes. Every word put on paper should be selected with this in mind.

- Simple and direct statements in plain English should be your goal - include enough groundwork and details to enable another engineer or geologist to evaluate your work. The test for adequacy is met when a qualified reviewer can follow your train of thought and your work procedures, and arrive at the same conclusion.

Keill's
References

PLACER DEPOSIT REFERENCES

1981, Standard Procedures for Sampling and Sample Preparation in Land Assessment, U.S. Department of Interior, Bureau of Mines, 40 pages.

Anatomy of a Mine from Prospect to Production, 1977, U.S. Department of Agriculture, Forest Service General Technical Report INT-35, 69 pages.

Boericke, William F., 1936, Prospecting and Operating Small Gold Placers, John Wiley & Sons, Inc., 145 pages.

Cook, Donald J., 1983, Placer Mining in Alaska, University of Alaska, MIRL Report No. 65, 157 pages.

Evans, James R., 1983, Placer Gold Sampling, Sample Processing and Evaluation of Results, U.S. Department of Interior, Bureau of Land Management, California State Office, 33 pages.

Evans, J.R., R.M. Waiwood and R.E. Reid, 1986, Procedures for Determining Discovery on Small Gold Mining Claims in California, in California Geology, October 1986, pages 219-228.

Haskins, Roger A. and Frederic B. Mullin, 1985, Handbook for Mineral Examiners, U.S. Department of Interior, Bureau of Land Management, H-3890-1.

Jenkins, Olaf P., 19964, Reprint of 8 articles on the study of placers, California Division of Mines and Geology, Special Publication 35, 60 pages.

Macdonald, Eoin H., 1983, Alluvial Mining: The Geology, Technology and Economics of Placers, Chapman and Hall, 508 pages.

Magyar, Michael J., 1987, Cost Estimating for Dewatering Alaskan Placer Effluents with PEO, U.S. Department of Interior, Bureau of Mines, Open File Report 39-87, 14 pages.

McCracken, David 1981, Advanced Dredging Techniques, Keene Industries, 240 pages.

McKinstry, Hough Exton, 1948, Mining Geology, Prentice-Hall, Inc., 680 pages.

Parkhurst, Dave W., 1987, 1988, Alluvial Placer Deposits, California Mining Journal, May, June, July August, September, October 1987; May, 1988.

Parks, Roland, D., 1957, 5th printing 1973, Mine Examination and Valuation, Addison-Wesley Publishing Co., pages 30-73.

Popoff, Constatine, 1966, Computing Reserves of Mineral Deposits: Principles and Conventional Methods, U.S. Department of Interior, Bureau of Mines Information Circular 8283. 113 pages.

Sahinen, U.M., et.al., 1964, Handbook for Small Mining Enterprises in Montana, Montana Bureau of Mines and Geology, Bulletin 39, 220 pages.

Schumacher, Otto L., 1984, Mining Cost Service, A subscription Cost Data Update Service, Western Mine Engineering.

Silva, Michael, 1986, Placer Gold Recovery Methods, California Department of Conservation, Division of Mines and Geology, Special Publication 87, 32 pages.

Stebbins, Scott A., 1987, Cost Estimation Handbook for Small Placer Mines, U.S. Department of Interior, Bureau of Mines, Information Circular 1970, 94 pages.

Stout, K.S., et.al., 1967, Mining Methods and Equipment Illustrated, Montana Bureau of Mines and Geology, Bulletin 63, 97 pages.

Stout, K.S., 1984, The Profitable Small Mine, Prospecting to Operation, McGraw-Hill, Inc., pages 83-98.

Thornton, Matt, 1979, Dredging for Gold, Keene Industries, 242 pages.

Wells, John H., 1969, Placer Examination, Principles and Practices, U.S. Department of Interior, Bureau of Land Management, 209 pages.

Wilson, Eldred D., 1961, Gold Placers and Placering in Arizona, Arizona Bureau of Geology and Mineral Technology Bulletin 168, pages 87-120.

Wolff, Ernest, 1969, 3rd printing 1980, Handbook for the Alaskan Prospector, University of Alaska, pages 171-320.

Additional Information

CARRYING POWER OF WATER*

Carrying power of water varies greatly with velocity, The following table indicates the duty of running water at various velocities:

30 feet per minute	-	lifts fine sand
45 feet per minute	-	moves fine gravel
120 feet per minute	-	moves 1-inch pebbles
200 feet per minute	-	moves 2-3 inch pebbles
320 feet per minute	-	moves 3-4 inch boulders
400 feet per minute	-	moves 6-8 inch boulders

*Source: Elements of Mining by George J. Young

SLUICE HEAD*

Sluice head is a rather indefinite term used in placer mining to describe the volume of water needed to separate the gold from the gravel in a sluice box. The average velocity of water in sluices of 6-inch drop to the 12-foot box is 140 to 400 feet per minute, which would correspond to a flow of 500 to 800 gallons per minute, depending on the depth of water in the box. Frequently it is convenient to speak of percent grade, instead of inches per foot, or per 12-foot box. The following table permits conversion:

Slope of 12-foot sluice box

Inches	Inches per foot	Percent
6	1\2	4.16
8	2\3	5.55
10	5\16	7.0
15	1 1\2	15.

*Source:Operating Small Gold Placers
by William F. Boericke.

BLACK SAND VALUES

Black sand is a general term applied to heavy, dark-colored mineral grains associated with placer deposits.

Over the years there have been many reports of black sands that allegedly contain gold and platinum in colloidal form or in a complex form not susceptible to recovery by conventional methods. Many special devices and recovery schemes have been promoted for extracting these alleged "hidden or locked" values. The fact that none of these methods has developed into self-sustained commercial operations is significant.

Where black and concentrates do contain gold it is generally present as fine-sized particles, some which may be coated by iron or manganese oxide. This "rusty" coating may be abraded or removed by tumbling or treatment wit nitric acid prior to being amalgamated to recover the gold.

Mining companies have found that from an economic stand-point, any gold tha cannot be recovered from the black sand by simple scouring and amalgamatior is not worth saving.

THIS PAGE
INTENTIONALLY
BLANK

Gold Distribution in Dry Placers

by Dave W. Parkhurst

The distribution of gold particles in all types of placer deposits is dependent upon a number of geological conditions existent at the time each deposit is formed and also upon subsequent alteration caused by changes in those conditions. However, there are some basic characteristics that are typical of each type of placer deposit, and these characteristics can be used by prospectors in the determination of where gold concentrations are most likely to be discovered.

The distribution of gold particles in dry placers is normally quite different than that which is found in the average stream or river placer, excepting some of the more recent deposits under similar conditions and placers formed during geological time periods when the climate was more conducive to the formation of stream placers.

Most placer gravels deposited ·der arid conditions are distin- ıshed by certain characteristics that · be easily recognized, even in ·e areas that presently have a ·uch wetter climate. Desert placer gravels are generally composed of predominantly angular materials, the bulk of which shows very little, if any, rounding or smoothing from the friction caused by fairly continuous water transportation and sorting. Because of the lack of a relatively constant water flow and the overall roughness of the placer gravels, gold particles usually do not work their way downward to bedrock or clay layers as rapidly as they would in a steadily-flowing stream. Since the particles are suspended in the materials at higher levels, the gold is much more accessible to secondary movement by subsequent surges of high water runoff.

Due to the turbulence and tremendous cutting power associated with sudden surges of heavy water flows, most of the gold particles are frequently redistributed throughout large volumes of thin-bedded gravels. ·ess the gravels are subjected to ·equent, extended periods of fairly ·y stream flows, the heavier ·allic particles have very little opportunity to be sorted and concentrated into confined gravel deposits. As a result, most desert placers tend to contain a fairly even distribution of very fine, small and medium size gold particles mixed in the gravel mass.

It is not uncommon to find the larger and heavier pieces of gold near the surface, with most of the smaller particles being deposited at a much lower level in the gravels. This condition results from the fact that the smaller particles of gold can work their way downward much more easily than the larger and coarser gold. However, the situation can be reversed when the larger gold particles were eroded first or during periods of maximum water flow and deeper erosion.

Another product of intermittant surges of heavy water flow over dry gravels is the formation of multiple, thin-bedded layers within the gravel mass. These layers are usually much more spread out and composed of a greater mixture of materials (soils, sand, pebbles and larger rocks) than those occurring in the stratified gravel structures normally found in most stream-deposited gravels. In many of the desert areas, these layers can be separated by narrow seams of clays or "caliche" (cemented gravels) that have formed during extended dry periods.

Depending upon the geological time period in which the heavier metals eroded, some of the gravel layers may contain gold while other lay-

ers do not. In certain instances, the gold-bearing gravel layers are separated by layers of barren gravels. Some placer deposits have been found that contain several barren upper layers separated by compacted clays overlying several gold-bearing layers, with barren gravels again being found at the bottom of the deposit. In some other instances, the layering is reversed or occurs in a different combination.

In most stream placer deposits, the constant vibration of the water-saturated gravels by constant water movement tends to settle the heavier and more resistant metallic particles toward the bottom of the channel. This process is very similar to the action of a shaker table or the movement of a gold pan. In desert areas, however, the intermittant surges of water flow are usually present for very short periods of time, such as immediately following a thunderstorm, cloudburst or rapid spring snowmelt. The flow of water is of such a brief duration that the underlying gravels are not softened or saturated with moisture. Because of this, the vibration or jigging action in the gravels by water movement is limited to the top layer in the gravel mass.

Under these conditions, the larger gold particles are deposited in fairly narrow streaks of limited length and depth, while the smaller particles are distributed throughout the surface gravels. Most "pay streaks" tend to be very irregular in occurrence and they can only be discovered by hit-or-miss prospecting efforts. Because of this irregular pattern of distribution, the use of prospecting techniques that apply to areas having a more constant stream flow are most likely to be more of a hindrance than a help when prospecting for desert placers.

The sporadic nature of gold deposition in dry placers also tends to create layers within layers, where some of the stratified gravels within a zone of deposition (usually defined by bands of clay or caliche) are gold-bearing while other strata in the same layer are not. Fairly narrow bands of gold-bearing gravels are sometimes sandwiched between barren bands of gravel. Some of these thin layers have been found to be very rich in gold content and they could be very easily missed by the unwary prospector. The sampling of any dry placer deposit should always be taken from the top to the bottom of the gravel mass and, if any gold is found, each individual gravel layer, seam, strata or channel should be sampled.

There are two more important factors that affect the distribution of gold in dry placer deposits, and these are the relative size and shape of the gold particles. Because of the limited water action in most desert areas, there is minimal abrasive action to grind, polish, smooth or flatten the pieces of gold. As a result, most of the gold particles are very rough and coarse. When coupled with the angular nature of the gravels, coarse and rough gold particles will generally not settle toward the bottom of the deposit. They will normally remain suspended in the materials and, in some cases, be distributed erratically throughout the entire gravel mass.

With much higher prices, it is likely that a number of dry placers discovered by old-time prospectors could now be mined at a profit.

As mentioned above, many of the smaller and more rounded gold particles will settle much more rapidly through angular gravels than will the larger and heavier peices. Due to their greatly reduced specific surface area, the smaller gold particles have much less friction with the rough and angular materials in the gravel than will the larger and coarser pieces. Extremely fine gold would remain thoroughly mixed in the gravels and would also travel long distances on the surface because of its ability to "float" on the surface of water.

Water does not adhere to the surface of gold, so the very fine and very thin gold particles will be supported by the surface tension of water. "Float" gold will often be transported for many miles from its source, and it will subsequently be moved even further with each new surge of water.

The nature of gold distribution in dry placers creates ideal conditions for the formation of large, fairly low-grade, placer deposits. It is also conducive to the formation of hidc placer deposits, which may be bur under a few feet to several hund feet of overburden, as well as the f mation of fairly rich, localized pock and streaks of gold-bearin ve Most of these desert place ill somewhat difficult to find and invol very careful and extensive sampling determine their value.

With much higher gold prices, it likely that a number of dry placers d covered by the old-time prospectc could now be mined at a profit. The is abundant evidence of older c placer mining activity in many areas the West, and these locations cou be well worth new prospecting effor In addition, some of the original plac miners used prospecting methoc that were learned in the wet plac areas and, as a result, they might hav missed some very good placer depo: its. Some of the older dry placer mir ing districts are now being reworke on a large scale, and new discoverie are being made on a regular basis.

The extremely large, relativel unprospected arid regions in th Western U.S. undoubtedly still contai several major gold placers that are sti awaiting discovery at some re date. Everything considered, th s a tremendous potential for the discov ery of new dry placer deposits. □

"While 'experts' seldom solve your problems—they will often explain your failures." ***Murphy Was A Miner***

PLACER MINING FOR GOLD USING SMALL COMMERCIAL PLANTS

by Ezra "Oz" Rosenbaum
Technical Director
Southwest Mining and Minerals

There are numerous areas in North America that have great potential for small to medium size placer mining. The writer has tested or developed placer mines from Alaska through Central America ranging up to 2,000 yards per day. Significant improvements in technology have made many more areas mineable. Additionally the price of gold which appears to have stabilized in the area of $300 has made placer mining attractive.

The principle underlying placer mining has not changed since prehistoric times. Given particles of equal size, the denser particle will settle faster in a fluid bed. This enables gold to be separated from lighter substances. The devices for accomplishing this range from the simplest pan to sophisticated centrifugal concentrators. This article will not only touch on the equipment available for placer mining, but just as important and before mining equipment is even discussed, the economics of placer mining.

Economics of Placer Mining

A 200 ton/day plant can process alluvial sand, gravel and cobbles at $3.00 to $4.00/ton. Variations in cost are dependent on several factors discussed later in greater detail. It is apparent that the average ore processed must have gold content in excess of $4.00/ton. Assuming gold at $10.00/gram ($315/T. oz) and allowing for considerable variations in placer ore tenor, one can envision a successful operation at 1.0 gram/ton (+/- 0.03 T. oz/ton). It would appear that we have left ourselves a wide margin of safety, and that considerable profit could be generated operating costs of $4.00/ton. But the only certainty in placer mining is the extreme variability of the feed stock.

The Mining Claim

Years of experience has taught the writer to favor placer claims exceeding 480 acres. In spite of what seemed like good test results, extensive excavation at one point on the site would suddenly cease to yield economic quantities of gold. A move of several hundred yards could again produce good results. Larger claim areas will often prolong the life of a placer mining operation.

Testing the Claim

Unlike lode mining, do not place too much reliance on test drilling. The extreme variability of placers is notorious. If small exploration test pits reveal the presence of gold in economic amounts, one should proceed to trenching on a pilot scale. It has been our experience that you can obtain more reliable information at lesser costs by trenching than drilling. A pilot operation can either be conducted by the claim owner or by contract with a reliable experienced operator. A word of caution—while superficial testing can be indicative of a potential valuable deposit, it is only an indication!

Availability of Water

Assuming that you have decided on the size of the operation you would like to run, the limiting factor might well be your water supply. In order to achieve a fluid bed condition, three tons of water are required for every ton of ore processed through the gravity separation unit. A two hundred ton-a-day plant will require 600 tons of water or 144,000 gallons per day. Some of this water will be re-used. Obviously the amount recovered will depend on the climate, the type of sediments and the rate at which they settle and the amount of the moisture retained by the tailings. Do not be optimistic about water recovery as a 50% loss is not unusual. If water is to be diverted from a stream, an inpoundment dam may be necessary which may add considerably to your costs. Water is vital to a placer operation. The success or failure of a placer may well depend on adequate supply and well-planned use of available water. Before committing money to a placer mine the supply of water must be assured.

If the property one intends to mine meets the criteria established, a plant using the following equipment could easily process 200 tons per day:

1. Grizzly 6' x 8' using 120 lb. rails with 2 inch opening.
2. Hopper feeder, vibratory type, placed directly under the grizzly.
3. Belt to 2 or 3 deck wet screen.
4. Wet screen 4' x 6' vibratory or shaker type. Final screen size 5/16 inch.
5. Centrifugal concentrator There are many models on the market. A unit capable of processing 15 tons/hour is required.
6. Classifier.
7. Spiral finishing unit—many models available.
8. Two water pumps, 250 GPM at 75' Head.
9. Piping and electrical—for longer term operations we suggest running on a diesel generator. 35 KVA is sufficient.
10. Lock boxes, scales, etc.
11. Optional methods of water recovery such as sand screws, etc., should be considered.
12. Rather than purchasing earth moving equipment initially, we suggest leasing or leasing with option to buy the following: front end loader, back hoe, bulldozer with ripper.

The above plant without earth moving equipment can be assembled for less than $75,000.

Operation of this plant is simple. The gravel and stone is fed to the grizzly (1) either by front end loader or dump truck, all material less than 2 inches passes through to hopper feeder (2). The discharged larger stones are pushed aside by the bulldozer for return later to the excavated area. The hopper feeder is set to deliver 20 to 30 tons per hour to the belt (3) leading to the wet screen. At the wet screen (4) water is supplied at the rate of not less than 3 tons/hour of material. The larger than 5/16" size gravel is fed by chute to a water recovery area. The less than 5/16" size is fed to the centrifugal concentrator. The centrifugal concentrator (5) is set to operate at a speed which will retain the denser particles. Both the speed of rotation and the amount of water must be optimized to obtain the best results. The addition of small amounts of surfactants (and sometimes ammonia) will improve results. When the con-

centrator becomes "loaded" with the heavier particles, clean up takes place by slowing the concentrator and discharging to lock box (10). The tailings are of course being continuously discharged through a chute to either a dewatering device or a pad. Depending on economic factors the tailings might be further processed by heap leaching or sent to a tailings area with no further processing.

The concentrate in the lock box varies enormously (from 100 to 5,000 T. oz to the ton of gold is not unusual). In order to obtain good gold recoveries in the next step, the concentrate must be classified by size and run separately in a spiral concentrator. A Sweco or similar classifier can be employed to cut the concentrate into four ranges: (1) retained 8 mesh; (2) retained 40 mesh; (3) retained 80 mesh; (4) passed 80 mesh. Each fraction is run through a spiral concentrator (7) such as a milspex or equal. The gold from fractions 2, 3 and 4 are combined and smelted into miners bars. Fraction 1 may contain some nuggets large enough for "jewelry gold". However, the writer believes too much time and attention is spent obtaining small amounts of nuggets. Improving processing techniques to obtain greater yields of "flour gold" will be more financially rewarding. The tailings from the spiral concentrator are sometimes re-run through the centrifugal concentrator (5) or extracted with cyanide.

The quantity of ore that can be processed through the equipment described is in a large measure determined by screen analysis. Thus, if 200 tons of ore is classified by screen size and only 10% passes a 5/16" screen, the centrifugal concentrator (5) will be called upon to handle 20 tons. Since its capacity is 15 tons/hour it could theoretically handle 120 tons in 8 hours. At 10% pass through the concentrator could handle 1,200 tons of ore per day! Similarly, if the grizzly (1) eliminates 50% of the ore, the wet screen (4) would be called upon in a two hundred ton a day plant to handle 100 tons per day. A 4' x 6' screen has a theoretical capacity to handle 10 ton/sq. ft./8 hour day. The screen could therefore handle 240 tons per day or at 50% through the grizzly, 480 tons of raw ore.

If one assumes a 100% pass through, i.e., all raw feed less than 5/16", the limiting factor would be the capacity of the centrifugal concentrator (5). In practice, seldom does the amount passed through the 5/16" screen exceed 40%. The centrifugal concentrator could easily handle 200 tons/day of raw feed.

Occasionally, the concentrate from the centrifugal extractor can not be further cleaned nor concentrated in a spiral. It therefore becomes necessary to send these "concentrates" to a refiner or smelter. At this juncture a digression is in order. The term concentrate is very loose. Webster's defines concentrate "To increase in strength by removing an admixed material so as to concentrate ores by washing." For practical purposes, this definition should be modified to include a statement that the concentrate must be in a form where it can now be processed economically. Thus if one starts with an ore containing 0.1 gram gold/ton of alluvial gravel and concentrates it ten fold, you would have a "concentrate" but *not* an economically viable product.

The concentrate produced by placering must be of sufficient value so that it can be transported to a mill, smelter or refiner and processed so as to turn a profit to the miner. The ordinary black sands produced in placering cannot be smelted directly. Let us assume that you have started with an ore containing 1 gram of gold per ton. You have done a good job of initial concentration—you have concentrated 100 to 1 and you now have 100 grams of gold/ton (3.2 T. oz). Can you smelt this directly into 3.2 T. oz of gold? Yes, but it will be trapped in 29, 162.8 T. oz of slag! Your gold would be no more available than before smelting. Can amalgamation be used to separate gold from black sand? Yes and no; some black sands where the particles of gold are not coated with silicious material are amenable to amalgamation. But the use of mercury is being restricted on BLM and Forest Service land and it can be hazardous to the operators.

Can gold in black sand concentrate be removed by cyaniding? Yes, in some cases, provided the gold is in an accessible form. In short, when is a "concentrate" not a concentrate? When the cost of producing the "concentrate" (placering) plus the cost of refining exceeds the value of contained gold.

Placering lends itself to increased production at nominal costs. Thus, an analysis of flow may reveal a point where a small amount of money spent can produce significant increases in production. It is not unusual to double production spending an additional $25,000. The returns for capital invested can be extraordinary and the time frame for getting these returns can be minimal. But a word of caution—know your property. Don't jump in without doing your homework. If you are not trained, hire experienced competant operators and be prepared to spend time on site. □

3-1

Gold analyses — myths, frauds and truths

W.G. Bacon, Bacon, Donaldson & Associates Ltd.
G.W. Hawthorn, Gary W. Hawthorn, P.Eng.
G.W. Poling, Mining and Mineral Process Engineering
The University of British Columbia

ABSTRACT

"Unassayable gold and platinum group metals" have come into vogue in the 70s and 80s in certain jurisdictions as a means of perpetrating fraud. Usual arguments are that a particular ore is not amenable to "conventional fire assaying". Explanations for unassayable gold usually revolve around: evaporation of micron-size gold; vaporization of organic gold complexes; volatization of gold halides; alloying of gold with PGM's which prevents fusion or alloying which prevents collection.

This paper reviews several of the myths and truths of gold and PGM assaying with the knowledge that not a single mine operates in the free world producing gold from unassayable ore.

INTRODUCTION

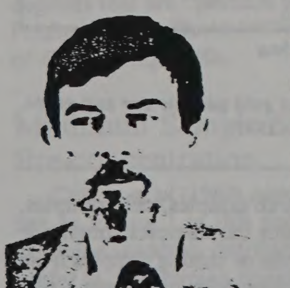
Accurate analyses of exploration data and estimation of ore reserves are two of the most important functions of professional geologists and engineers in mine development. The problems encountered become particularly acute when dealing with precious metal values where very low grade deposits (in the range of 0.01 Troy oz per ton or ~0.2 ppm (g/t) by weight) sometimes are economic or constitute "ore". To emphasize this "scarcity effect", a gold assay of 0.4 g/t (0.01 T oz/st) would result from the presence of a single 420 μm (35 mesh) gold flake (which weighs 1 milligram) in a 5 kilogram siliceous sample.

The current high gold price and the prospects of high profit margins awaiting exploitation of even very low grade gold deposits, has spawned a sizeable group of incompetent or fraudulent gold assayers. These "assayers" will often report significant precious metal values where no such values occur. Failures of legitimate assay labs to confirm these "significant" assays are often attributed, by the charlatan, to their capability of detecting "unassayable gold" while the check assayer cannot. Geologists and engineers lacking detailed knowledge of sampling and assaying techniques are often hard-pressed to convince a gold-fever aroused company executive or investor-client to abandon a property on which attention-grabbling values have been reported, often by more than one laboratory. Professionals engaged in such evaluations must acquire an understanding of contemporary precious metal sampling and assaying techniques. Only then can they determine whether misrepresentation of values have been perpetrated.

The authors have independently investigated over 30 prospects and as many "assay" labs in which significant precious metal (gold, silver and platinum group elements) values have been reported when in fact near-zero values were present. We write this paper, with emphasis on fire assay techniques which are a well recognized analytical standard, or preconcentration technique in the hope that the industry will learn to

Keywords: Mineral processing, Gold analyses, Assays.

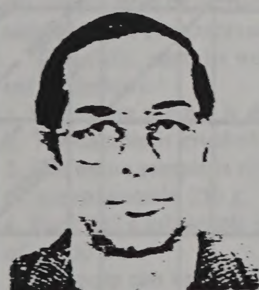
Paper reviewed and approved for publication by the Canadian Mineral Processors Division of CIM.



W. Gordon Bacon received his B.A.Sc. degree in mineral engineering and his Ph.D. degree in metallurgical engineering from The University of British Columbia. As an undergraduate he worked during the summers as a test engineer on flotation reagents and copper leach extraction. After graduation, he worked in Saskatchewan for three years with International Minerals and Chemicals Corp. before returning to The University of British Columbia for his doctoral studies.

In 1972, he became a principal in the metallurgical consulting firm of Bacon, Donaldson and Associates Ltd., of which he is now president. Since 1982, Dr. Bacon has also served as adjunct assistant professor in the departments of Metallurgical and Mineral Engineering.

Dr. Bacon is a member of The Canadian Institute of Mining and Metallurgy, the Association of Professional Engineers of British Columbia, American Society for Metals, National Association of Corrosion Engineers and Electrochemical Society and a Fellow of the Institution of Mining and Metallurgy, England.



Gary W. Hawthorn, graduated from Queen's University in 1964 with a B.Sc. in mining engineering. From then until 1982, he worked first for Cominco Limited and later for Placer Development Limited, principally as a mill superintendent. He has been a mill superintendent at Con Mine, Craigmont, Marcopper, Equity Silver and Pinchi Lake, the latter two on start-up. In 1982, he opened a consulting mineral processing practice in Vancouver, British Columbia, and in 1988 formed Westcoast Mineral Testing Inc., of which he is president.

George W. Poling obtained his B.Sc., M.Sc. and Ph.D. degrees from the University of Alberta in 1957, 1961 and 1963, respectively. He worked from 1963 to 1968 as a senior and research metallurgical engineer studying the surface chemistry of corrosion inhibition at the Texaco Research Centre in Beacon, New York. Dr. Poling has been professor of Mineral Process Engineering at The University of British Columbia since 1968 and has published over 150 papers dealing with froth flotation, solid/liquid separation, tailing disposal, corrosion inhibition and infrared spectroscopic studies of solids surfaces. Dr. Poling is currently on two years' leave of absence from the Department of Mining and Mineral Process Engineering at The University of British Columbia to serve as coordinator of applied research for the Mining Association of British Columbia. He is a Professional Engineer of British Columbia, a member of CIM and belongs to the IMM (London) and AIME.

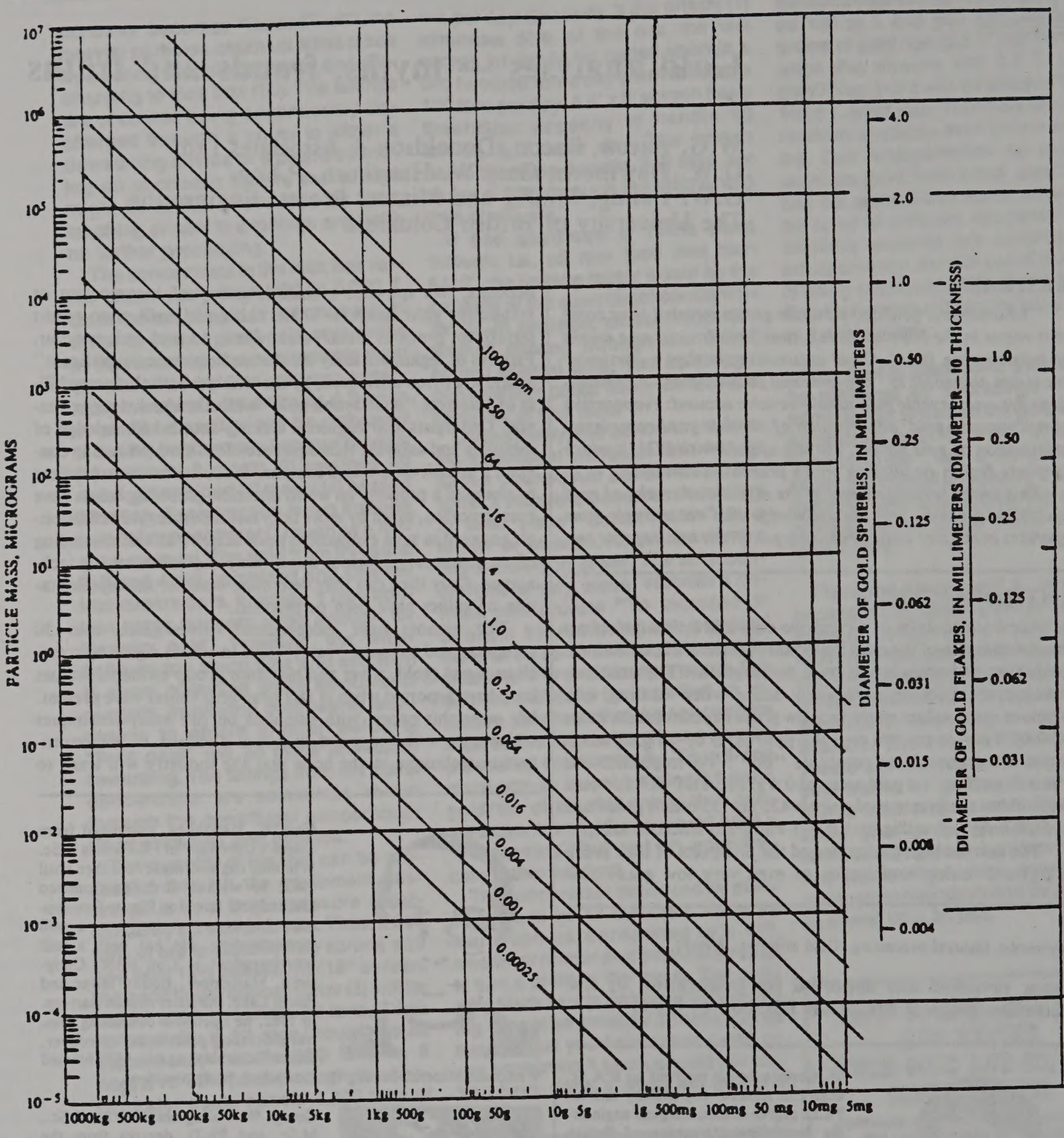


FIGURE 1. Size of sample required to contain an expected 20 particles of gold as a function of the combination of gold particle size and grade, assuming all gold particles to be of uniform size and randomly distributed in the deposit (after Clifton *et al.* . 1969).

recognize and then ignore fraudulent assayers. At present, the prospects of seeing them out of business, or better yet behind bars, seem improbable.

This paper includes discussion of sample preparation, fire assay procedures, other noble metal analytical procedures and typical myths that have been used to justify fraudulent assay procedures to find "unassayable" gold.

Sampling of Gold Ores

Methods of Obtaining Samples

A wide variety of techniques have been used to obtain samples of gold ore. These include grab samples, chip samples, channel samples, panel samples, mine car samples, muckpile samples, core samples or sludge samples or cutting samples from diamond drills, rotary drills, percussion drills or reverse circulation drills, bulk samples taken by backhoes or bulldozers or by drilling and blasting. The application of these various sampling techniques to placer gold deposits is described in Macdonald (1983). Sampling techniques for lode gold deposits are described in McKinstry (1948), CIM Special Volume 9 (1968), Jones (1974) and Metz (1985). A comparatively recent innovation in hard-rock sampling technique is the use of a portable diamond-impregnated circular saw to cut the edges of

TABLE 1. Gold particle content of various sample sizes as a function of gold particle size (cubes) for a 1.56 g/st (0.05 oz/st) ore (after McLean [1982])

Gold size µm	Mesh	No. of Au particles per (10 lb) 4535 g sample	Number of Au particles per assay sample: 1 AT*	2 AT	5 AT	1000 g (33 AT)	2000 g (66 AT)	10 000 g (330 AT)
1650	10	0.17	0	0	0	0	0	0.
833	20	0.71	0	0	0	0	0	1.6
589	28	2	0	0	0	0	1	
295	48	16	0	0	0.5	3	6	35
208	65	46	0.30	0.60	1.5	10	20	101
147	100	128	0.84	1.68	4.20	28	56	281
104	150	370	2.4	4.8	12.0	81	163	368
74	200	1000	6.6	13.2	33.0	220	440	2193
45	325	4588	30.4	60.8	91.2	1011	2022	
38	400	7959	52.6	105	263	1736	3472	
20		49920	330	660	1650	10890	21780	
5		3276000	21671	43342	108355			
2		50000000	330000	660000				

* AT = Assay Ton

channel samples so that the sample volume is more regular and uniform, Magri and McKenna (1986).

Size of Samples

Relatively simple methods to calculate adequate sample size have been presented by Gy (1968, 1974, 1979), Clifton *et al.* (1969), Visman (1969) and Ingamells (1980). In sampling gold deposits containing relatively coarse gold, these techniques will all dictate that sometimes alarmingly large sample weights are required to obtain a representative sample, particularly of a low grade deposit. For example, a gold deposit containing 0.16 g/t (0.005 T oz/st) (250 mgm Au/m^3) with the coarsest gold being 1 mm diameter, would require a minimum sample size of 1.5 m^3 (or around 5000 lb) to achieve an accuracy of ±25 mgm/m^3 at a 95% confidence limit. A general conclusion of Clifton's work is that "the number of gold particles in the sample is the only factor controlling the precision of chemical analyses". A precision of ±50% is achieved at 95% certainty when samples for analyses each contain a minimum of 20 particles of gold (or ±20% @ 67% certainty). Although these relatively simple sampling theories depend on the gold particles being randomly distributed (which seldom occurs) they certainly provide very useful indications of adequate sample size. Implications of these sampling statistics have led at least one author to conclude that there are deficiencies in sampling placer gold deposits that are "perhaps insurmountable at reasonable cost" Fricker (1976). David (1977) gives a good review of the work of Gy and Ingamells.

Minimum Sample Size, Need for Pre-concentration

Clifton *et al* (1969) nomographs, one of which is reproduced as Figure 1 in this paper, are based on simple statistics assuming randomly distributed gold. These nomographs are very useful in determining minimum sample weights and the possible need for physical preconcentration prior to analysis. Application of the Figure 1 nomograph should ensure that with 95% probability, the true gold content will be within ±50% of the gold content obtained by chemical or instrumental analysis of the sample. This degree of precision will be attained if the particular sample contains a minimum of 20 particles of gold. The original Clifton *et al.* paper provides additional conversion scales to provide sample sizes required for both higher and lower degrees of precision. Their paper also documents the high probability of certain samples mistakenly assaying zero gold content if fewer than five particles of gold are possible in any sample.

Table 1 tabulates the effect of gold particle size and the number of particles in a 4.5 kg (10 lb) sample and in other various assay sample sizes (weights). The implication is that small sample sizes or large gold particle size will invalidate an assay of small samples.

Table 2 tabulates the nugget effect for one particle of gold of a given size on an assay of a sample of given weight.

Preparation of Samples for Assay

Once a representative sample has been taken in the field, the sample must be prepared in the laboratory to produce a subsample that is both representative and of suitable size for assay.

Generally, samples weighing less than 10 kg will be processed as follows:

1. The entire sample is dried.
2. The sample is crushed to −6 mm.
3. The entire sample is riffle split to obtain one or more splits of approximately 250 g.
4. Each 250 g subsample is pulverized to produce an assay pulp of −150 µm.
5. The pulp can be rolled and sampled or better yet, obtained by using a micro riffle splitter.

Care must be taken to prevent cross contamination of samples during crushing and pulverizing. When metallic gold is present, disc pulverization often leads to severe cross-contamination problems. Ring and puck pulverizers should then be used, Merks (1985).

Reference to Figure 1 (after Clifton *et al.*) will indicate that low grade samples containing relatively coarse gold cannot be represented by a pulverized subsample of a 250 g split. One partial solution is to add a "metallics assay". Following pulverization to −150 µm, the pulp is screened at −150 µm. Because gold is not readily pulverized, coarse gold will be retained on the −150 µm mesh screen. The entire material retained on this screen is weighed and assayed separately. Screen undersize is sampled and assayed separately. Total gold contained in the original sample is calculated from the combined weights and assays of the "metallics" and the screen fines.

Another procedure, designed to ensure that coarse gold, readily liberated gold and refractory — encapsulated gold are

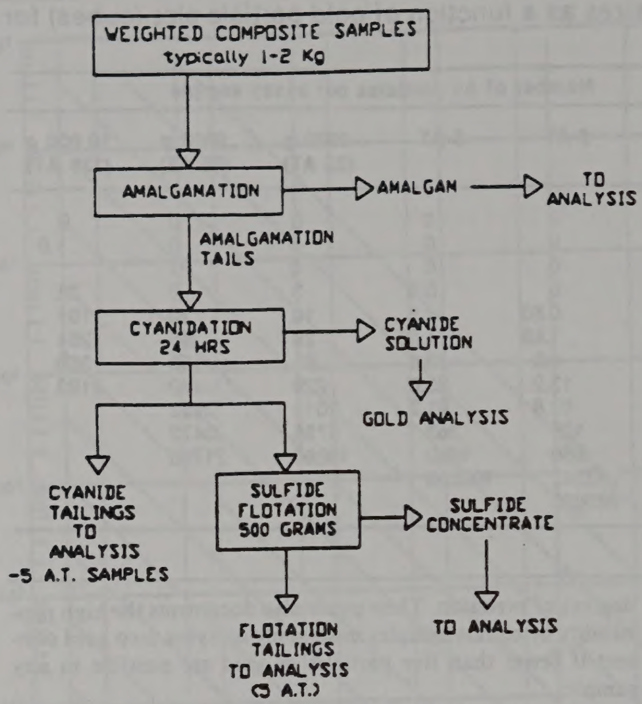


FIGURE 2. Composite gold extraction flowsheet (after McLean, 1982).

all included in assay results, is shown in Figure 2. This procedure, while costly, will not only give more accurate results but also indicates partial process requirements.

Fire Assaying for Gold Silver and Platinum Group Metals

Any modern review of the analytical chemistry of the noble metals must place particular emphasis on the fire assay. The fire assay is the most favoured noble metal analytical technique both in history and at present. In the last decade or so, instrumental techniques have become more popular and in combination with the fire assay method have supplanted the standard gravimetric finish-fire assay in many cases.

The classical lead crucible fusion assay is the most common and successful procedure for the concentration and collection of the noble metals.

One basic reason for the continued use of the fire assay technique is the relatively large sample size that can be treated by the technique, usually one assay ton or 29.1667 g. Most instrumental techniques do not allow sample sizes larger than a few grams which are usually inadequate (Fig. 1). Another reason is that the fire assay technique is relatively forgiving and free of interferences.

The most common fire assay procedure for gold and silver analysis is:

1. mixing of the ore with the flux components plus a noble metal collector in a fire clay fusion pot (crucible);
2. fusion of the mixture;
3. pouring the melt into iron moulds;
4. separating the slag from the lead or other collector metal;
5. scorification when required;
6. cupelling the lead-collector button;
7. recovering a precious metals bead, weighing;
8. parting of the bead with nitric acid; and
9. gravimetric determination of Au and Ag separately.

The purpose of the crucible fusion and the flux components is to eliminate the gangue minerals and to concentrate the noble metals into lead, drastically reducing any interfering elements. The flux must fuse to form a slag at a relatively low, easily reached temperature. The flux forms the slag by dissolution of the sample components *not* by melting. The flux is usually composed of all or parts of:

Chemical constituent	Chemical formula
Lithage	PbO
Flour	(household flour)
Borax	$Na_2B_4O_7 \cdot 10H_2O$
Sodium carbonate	Na_2CO_3
Silica	SiO_2
Potassium nitrate	KNO_3

For most fire assays, a standard flux recipe is used. Under ideal conditions the flux is designed by the assayer after he knows the chemical and mineralogical composition of the sample. This can involve semiquantitative emission spectroscopy or inductively coupled plasma atomic emission spectroscopy and basic mineralogical analysis. An experienced assayer may simply use a small subsample, a low power microscope and an acid to perform his analysis. In the end he should determine if the sample to be assayed is neutral, reducing or oxidizing before he can design a successful flux for the assay of the sample. All sulphides and carbon are natural reducing agents. Many oxides are oxidizing agents.

Once the flux is selected it is placed in the fusion crucible with the pulverized sample and the charge is thoroughly mixed (a cover of borax or flux mixture is added to prevent dusting). The crucible is then placed in a fusion furnace at 1000°C to 1100°C until the fusion is complete (usually 30 to 60 minutes). After fusion, the crucible is removed from the furnace and the molten charge is poured into iron moulds. During the fusion process a portion of the PbO is reduced to metallic lead which becomes the "collector phase". The high density molten lead and precious metals sink to the bottom of the clay crucible and to the bottom of the iron mould.

When the fusion charge has cooled and solidified, the lead button is broken away from the slag. Normally the slag is discarded unless it is reassayed to check for suspended or "shotted" precious metals. The lead button, weighing 20 g to 30 g, is next hammered into a cube to remove any adhering of slag and to assist in handling with tongs during subsequent cupellation.

Throughout the crucible fusion portion of a fire assay an experienced assayer watches carefully for unusual behaviour, such as:

- boiling over of the molten slag;
- high viscosity on pouring;
- hang-up of lead on the crucible after pouring;
- shotting of the lead in the slag;
- matte formation on the lead;
- speiss formation;
- dirty lead;
- hard, brittle or non-ductile lead on hammering the cube;
- unusual texture to the lead;
- too much or too little lead in the button; and
- any unusual occurrence from pouring to hammering the lead cube.

Depending upon the severity of the problem the assayer may scorify (oxidizing fusion with an excess of granulated lead) the lead button or simply perform another fusion. A second fusion with the knowledge learned from the first improper fusion is the preferable route.

The next step in the classical fire assay is cupellation. At

TABLE 2. "Nugget effect" of gold particle sizes versus sample weight (after McLean [1982])

Gold size µm	Mesh	Wt. of one gold particle (mg)	Change in gold assay per particle of gold · T.O. Ton						
			1/2 AT*	1 AT	2 AT	5 AT	1000 g 32 AT	2000 g 64 AT	10 000 g 320 AT
1650	10	88	176	88	44	17.6	1.75	1.38	0.27
833	20	11	22	11	5.5	2.2	0.34	0.17	0.03
589	28	4	8	4	2	0.80	0.215	0.062	0.021
295	48	0.50	1.0	0.50	0.25	0.10	0.016	0.008	0.002
208	65	0.17	0.34	0.17	0.18	0.03	0.005	0.002	0.001
147	100	0.061	0.12	0.06	0.03	0.01	0.002	0.001	0.001
104	150	0.021	0.04	0.02	0.01	0.004	0.001	0.001	0.001
74	200	0.0078	0.02	0.01	0.00	0.002	0.001	0.001	0.001
45	325	0.0017	0.004	0.002	0.001	0.001	0.001	0.001	0.001
8	400	0.00100	0.002	0.001	<0.001	<0.001	<0.001	<0.001	<0.001
20		1.56E-4	0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
5		2.41E-6	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
2		1.56E-7	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001

Note: E-6 = 10^{-6}
*AT = Assay Ton

this stage a direct addition or "inquartation" of silver is sometimes made to facilitate possible subsequent acid-parting of the silver from the gold-silver bead (the inquartation is sometimes made in the fusion stage in high productivity-commercial assay labs). The purpose of cupellation is to extract the noble metals from the lead. This is accomplished by oxidizing the lead in a porous cupel made of MgO, bone ash or cement in a muffle furnace at 950°C to 1000°C. The oxidized lead (liquid litharge) is mainly absorbed into the cupel with approximately 2% of the lead being volatilized. As with the fusion stage, observations made during the cupellation procedure and of the resultant bead inform the assayer whether the cupellation has been successful. An example of difficulty is the occurrence of tellurides in a sample; this can lead to non-spherical multi-beads. These difficulties can be eliminated by repeat assays of preoxidized or roasted telluride ores. Cracks in the cupel and certain scoria indicate unsuccessful cupellation.

In the classical fire assay, the gold and silver contents of the Doré bead obtained from cupellation are determined gravimetrically. After carefully weighing (to ~.002 mg) the gold-silver Doré bead, the silver (and any base metal) contents are dissolved away in hot nitric acid (parting) and the remaining gold is reweighed. The silver content is obtained by the difference in weight. If additional silver is added as "inquartation" both the weight and purity of this silver must be assured in order to determine accurately the silver content of the sample. As the name implies, the optimum ratio of silver to gold during inquartation is approximately 4:1.

If present in a sample, platinum group elements (PGE) are also collected in the lead button during a crucible fusion. In some laboratories, nickel sulphide rather than lead is preferred as a fusion-collector for the platinum group elements, Dixon (1975). Upon cupellation, platinum, palladium and rhodium alloy with the gold-silver Doré bead and cause characteristic changes in its appearance. Nitric acid parting will dissolve most of the palladium, part of the platinum and none of the rhodium. Iridium does not alloy with the Doré while osmium and ruthenium tend to form volatile oxides which can be partially volatilized and lost during cupellation.

While procedures are available to complete analyses of gold-silver and the PGE using the classical fire assay and wet chemical analyses these are now often superseded by instrumental techniques. For Au-Ag-Pt-Pd-Rh assays, standard crucible fusion and cupellation are often followed by digestion of the bead and use of atomic absorption or emission spectroscopy to finish the analyses. Sometimes the Doré bead is analyzed directly using neutron activation analysis. To include the other PGE's the lead collection button is often digested and analyzed instrumentally.

Other Noble Metal Assay Methods

Several types of spectroscopy are utilized directly on solid samples with, or without physical preconcentration of the precious metals. Pyrometallurgical preconcentration using the fire assay procedures of crucible fusion plus lead collection alone or in combination with cupellation are now commonly used in combination with instrumental analyses.

Atomic Absorbtion Spectroscopy (AAS)

This is the most common instrumental technique used for completing the assay of noble metals. It is often used after preconcentration by fusion and cupellation. The use of the solvent extractant methyl isobutyl ketone (MIBK) further to concentrate the noble metal has improved its sensitivity and reduced interferences. Sometimes dissolution and MIBK extraction are used directly on ore samples. Modern AAS employs flameless spectrographs, various buffers or releasing agents and chemical extractions that have greatly reduced the interferences that have plagued AAS in the past. It is most important as the finishing step in noble metals assaying.

Emission Spectroscopy (ES)

This technique allows for simultaneous elemental analysis over a wider concentration range than AAS. Detection limits are too high to allow direct analyses of ores and thus preconcentration is necessary. ES is useful for the detection of trace elements with products such as precipitates from chemical separations or distillates and cupellation beads.

Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES)

The use of inductively coupled plasma emission spectroscopy has largely replaced gravimetric analysis of fire assay preconcentrated buttons or beads. The very high plasma temperatures reached in the plasma greatly reduce the interelement interferences. The greater range of concentration capability of ES is maintained as is the simultaneous multi-element capability. Analysis of cupellation beads by ICP-AES allows

assays in the ppb range. The high capital cost of ICP instruments (> $100,000) discourages widespread application of this technique.

Neutron Activation Spectroscopy (NAS)

The analytical chemistry of noble metals lends itself to NAS because of sensitivities that are at least two orders of magnitude more sensitive than other methods. However, extremely complex interferences do not allow such high sensitivities to be readily realized. Without preconcentration, the maximum sample size is approximately 2 g to 3 g and thus if there is inhomogeneity in the noble meal distribution the sample size precludes accuracy. Analysis of cupellation beads allows noble metal assays in the ppb range.

X-ray Fluorescence Spectroscopy (XRF)

Separation, recovery and/or preconcentration techniques are necessary because this method requires samples with at least 1 mg of the noble metal in the sample for accurate determination. This method is often used with fire assay preconcentration into a lead button followed by cupellation into a silver or gold bead. The bead is annealed, flattened and reannealed and analyzed as a solid. Portable XRF analyzers are sometimes used for analyzing gold distribution in stope faces or on drill cores or in drill holes but this technique is semi-quantitative at best.

Myths

Complex Stable Compounds

"Certain ores which contain precious metal values are not amenable to recovery or assay by standard methods. In our particular ore the standard lead collection fails to gather values. We believe the metals occur as complex and very stable compounds. These compounds dissolve and go into solution as the compounds. They likewise are not decomposed by the standard lead fusion assay but remain in the slag rather than gather in the lead."

This quote demonstrates a typical belief that "unassayable" gold is the result of complex stable compounds. This is, of course, incorrect as the stability of compounds is not a consideration if the flux is correctly designed. If a gangue mineral does not dissolve it becomes immediately apparent in the slag during pouring or removal of the solidified slag from the lead button. Complex stable compounds are easily dissolved when a suitable flux is incorporated in the fusion allowing the lead to collect any noble metals present.

Noble Metal Interferences

There is a widely held misconception among incompetent "assayers" that the presence of various noble metals causes interferences that negate the fire assay procedure. The following data are from the popular mining magazine *California Mining Journal* and was authored by Alvin C. Johnson, Jr., Ph.D. The article is titled, "Platinum Group Interference Phenomena in Fire Assaying Methods"and the data presented are shown in Table 3.

These data are from analyses of splits from an ore sample that is described as "precious element bearing and is from a refractory desert alluvial deposit that is located in southwestern Arizona. The deposit is quite extensive and does not respond well to standard fire assay methods or atomic absorption methods".

Johnson hypothesises that "the precious elements are not initially reduced to low valence or metals prior to or during analysis and that they do not react chemically or metallurgically in a normally predictable fashion." Experience suggests to Dr. Johnson the existence of numerous refractory precious element compounds which have not, as yet, been identified.

Our analysis of the above data (Table 3) would indicate that at least three of the reported "assays" are not correct values. To report gold results that vary from trace, 0.001, 0.01 to 0.79 oz/st is unacceptable and indicates human incompetence and not errors in the method. The presence of platinum group metals does not affect the results of trace assay methods by competent assayers. Because the D method of Table 3 utilized a crucible fusion with the precious metals collected in a lead button, the probable errors would involve manipulation of the emission spectroscopic data.

Micrometre Gold

Micrometre-sized gold is often blamed for the inability of the classical fire assay to result in the correct assay value. This makes no sense since micrometre-sized gold particles are ideal from both a sampling and assaying perspective (Fig. 1). Truly micrometre-sized gold should result in the most reproducible assays from all assay techniques.

Sometimes the micrometre-sized nature of the gold is stated to "cause volatilization of the gold during fire assay". This is also nonsense as the fire assay fusion does not exceed 1100°C and the boiling point of gold is 2600°C.

Myths also abound that the micrometre-sized gold particles are not collected by the lead but stay suspended in the molten slag. Again this is untrue, with micrometre-sized particles the collision statistics with the lead collector are favoured and there should be even less chance of gold loss to the slag.

Volatilization of Gold Compounds

A common mistakenly-held theory is that unassayable gold occurs as a halide compound, the favourite of which is chloride. These incorrect theories postulate that the halide is volatile as a gold chloride. While there are two stable gold chloride compounds ($AuCl$ and $AuCl_3$) that can be made to volatilize and transport, this is only true in an atmosphere of chlorine. In a fire assay fusion, $AuCl$ and $AuCl_3$ would decompose to gold and chlorine gas at 170°C and 254°C, respectively. The gold would then collect in the lead as from any other gold source.

Similar incorrect theories abound for organo-complexes of gold and other low melting point compounds.

Corrected Assays

A common procedure used by incompetent assayers is the so called "corrected" assay technique. The "corrected assay" is a legitimate technique used to account for minor losses to the slag and cupel that inevitably occur. A legitimate corrected assay technique is sometimes used for an initial check of potential losses particularly when assaying high grade ores and bullion.

A legitimate corrected assay consists of a standard fire assay of the ore or bullion with fire reassays of the first fusion slag (refusion) and an assay of the cupel from the first assay. The total assay is then the total of the gold or noble metals from the three fire assays (first fusion, slag refusion and cupel fusion). For high grade ores or bullions this total of the three assays can be 2% to 3% higher than the first fire assay (Table 4, from Hosking [1982]).

The fraudulent assayer performs the "corrected" assay in the same way initially but continues the process by reassaying the second slag and second cupels. This may continue for as many as four times thus having assays for three or four assays

TABLE 3.

Element	Assay method A	B Assay	C oz/ton	D
Ag	0.8	0.01	0.1	0.29
Au	trace	0.001	0.01	0.79
Pt				1.31
Pd				0.32
Ir				1.81
Os				1.53
Rh				1.31
Ru				2.07

A = fire assay, reductive flux
B = fire assay, "standard" flux
C = AAS after aqua regia leach
D = ES of fire assay lead button

of slags and cupels for one sample. This type of "corrected" assay is easily recognized to be fraudulent if the amount of noble metal in any assay after the first is more than 5% of the noble metal recovered in the first assay. In some cases these "corrected" assays are claimed to recover 50% to 75% of the noble metal in the subsequent reassays of the slags and assays of the cupels. This is always fraudulent or incompetent and these assays should be ignored.

Other Fraudulent Assay Technologies

There are various other techniques used by fraudulent or incompetent "assayers" in perpetuating their businesses. While the names of the techniques they use have legitimacy the techniques are either misapplied, overly complicated, useless, antiquated or simply not applicable to modern noble metal assaying.

Among the methods or techniques used to "assay" for "unassayable gold" that should alert the reader to possible fraudulent, or incompetent assays are:

- wet chemical analyses, while certainly legitimate, is not used for noble metal analysis or ore samples as a standard method;
- electrolysis methods;
- unusual time — temperature combinations during fire assay — they are unnecessary;
- basic or acidic pretreatment of ores prior to fire asssay — especially if both the lixiviant and undissolved solids need to be assayed; and
- methods that prevent "volatilization" of noble metals.

Evaluation of Laboratories

The best method to evaluate a laboratory is to consult with well recognized mining firms in the locale producing noble metals. These firms will have their own assays checked by outside commercial laboratories. These laboratories will be legitimate.

Secondly, submission of samples of the following nature to evaluate the laboratory is good practice:

- duplicate samples from another laboratory;
- blank samples; and
- known standard samples (USBM and CANMET).

Significant deviation from zero on the blank or from the standard value leads one to conclude the laboratory may not be of high calibre. Large deviation suggest incompetent or fraudulent assaying.

If the duplicate assays are different, then a third laboratory must be selected because either of the first two laboratories may be in error.

TABLE 4. Example of gold losses in fire assay (after Hosking [1982])

Method of loss	% Gold loss
to slag	0.2
to crucible	0.2
to hammering button	0.1
to cupel	0.7
to parting solution	0.03-0.8
total loss	1.15-2.0
gain by silver retention	1.25-1.65

Conclusions

The pyrometallurgical techniques of fire assaying, whether used for the gravimetric analyses of total gold and silver values or for preconcentration prior to instrumental analyses, remain the most reliable and most respected procedure for analysis of precious metal values. The procedure is relatively expensive, hence they are now often augmented by instrumental analyses for many routine high production-number assays.

Of all the possible errors in sample analyses, lack of representivity of the sample to be assayed often constitutes the greatest source of error. Nugget effects of coarse precious metal values necessitate strict attention to the adherence to using meaningful sample size or preconcentration techniques. Experienced competent assayers will find precious metals if present. The fire assay technique, while requiring competence and skill, is a relatively forgiving technique. The most respected expert in the field of fire assaying in this century, Professor F.E. Beamish, once stated "that during 40 years of research in the field, he had not experienced a single example of failure of the classical assay to find a paying ore". The authors of this paper have a collective experience with fire assay on the order of sixty years and unequivocably agree with Beamish! A major conclusion is that "there is no such thing as unassayable gold and silver".

REFERENCES

CIM Special Volume 9 (1968), Ore Reserve Estimation and Grade Control; Can. Inst. Min. and Met. p. 321, Montreal.
CLIFTON, H.E., HUNTER, R.E., SWANSON, F.J., and PHILLIPS R.L., (1969), Sample Size and Meaningful Gold Analysis. USGS Professional Papers, 623-627 A-F,C1-C15.
DAVID, M., (1977), Geostatistical Ore Reserve Estimation. Amsterdam. Elsevier. p. 364.
DIXON, K., JONES, E.A., RASMUSSEN, S., and ROBERT, R.V.D., (1975), The efficiency of the fire assay procedure with nickel sulphide as the collector in the determination of platinum, silver gold and iridum; Natn Inst. for Metallurgy, Report No 1714
FRICKER, A.G., (1976), Placer Gold Measurement and Recovery. Aus., IMM, Melbourne Branch, Sampling Symposium, Sept., pp *115-127.*
GY, P., (1968), Theory and Practice of Sampling Broken Ores; Ore Reserve Estimation and Grade Control, CIM, *pp. 5-10.*
GY, P., (1974), The Sampling of Broken Ores: A Review of Principle and Practice, in Geological, Mining and Metallurgical Sampling; London Institute of Mining and Metallurgy, *pp. 194-205.*
GY, P., (1979), Sampling of Particulate Materials, Theory and Practice; Amsterdam, Elsevier, p. 431.
HOSKING, J.W., (1982), Review of Analytical Methods for Determining Trace Amounts of Gold in Ores and Process Streams; Aus. IMM Carbon in Pulp Seminar, July. *pp. 351-367.*
INGAMELLS, C.O., (1978), General Sampling Theory; Int. Prec. inst., 1st Sym. on Sampling.
JOHNSON, A.C., (1985), Platinum Group Interference Phenomenon in Fire Assaying Methods, *California Mining Journal, pp. 2-3.*
JONES, M.J., (1974), Geological, Mining and Metallurgical Sampling; Inst. of Min. and Mertallurgy, London. p. 268.
MacDONALD, E.H., (1983), Alluvial Mining; The Geology,

Technology and Economics of Placers, London, Chapman and Hall Ltd., p. 508.
McKINSTRY, H.E., (1948), Mining Geology; Englewood Cliffs, Prentice-Hall Inc., p. 680.
McLEAN, DANIEL C., (1982), Practical Considerations in the Sampling and Estimation of Reserves in Low Grade Gold Ore Deposits; The International Precious Metals Institute International Seminar, London.
MAGRI, E.J., and McKENNA, P., (1986), A Geostatistical Study of Diamond-saw Sampling versus Chip Sampling; *Jour. of South Africa Inst. Min. and Met., Vol. 86, No. 8, pp. 335-347, Aug.*
MERKS, J.W., (1985), Sampling and Weighing of Bulk Solids; Trans Tech Publications, Germany.
METZ, R.A., (1985), Applied Mining Geology: Problems of Sampling and Grade Control; New York, Amer. Inst. of Mining, Metallurgical and Petroleum Engineers, p. 164.
VISMAN, J., (1969), A General Sampling Theory; *Mat. Res. Stds. 11 (8). p. 8.*

SAMPLING ERRORS AND THE RELIABILITY OF SAMPLING *

Sampling errors may result in the rejection of valuable properties or in the waste of large sums of money spent in developing uneconomic deposits. Some errors are common to all placer environments, others are specific to particular placers, some are errors of commission and some are errors in judgement.

Errors common to all placer environments

The reliability of sampling increases with the number of samples taken and,hence, with reductions in the volumes of influence assigned to each sample and increased reliability with increased sample volumes. There are economic considerations and while in both cases the ultimate sample has a volume to volume of influence ratio of 1:1 this, of course is quite impractical and at some point a compromise must be made between sample cost and reliability.

Incorrect sample dimensions: For each deposit there is an optimum density of sampling that is based upon the geology of the deposit and the selection of methods that will hold sampling errors within pre-determined limit. Consider two adjacent samples each representing the same volume of influence. One is a pit sample having a cross-sectional area of 1 m^2 and a sample interval of 1 m; the other is a borehole sample of a diameter of 10 cm and the same sample interval of 1 m.

Pit sample volume $= 1 \times 1 \times 1 = 10^6\ cm^3$
Bore sample volume $= (11/14) \times 10^2 = 7857\ cm^3$
volume ratio $= 127:1$

It is contended by some that the prime consideration in sampling is the number of samples taken rather than the size of individual samples. There are grounds for this view when sampling deposits of similar sized particles such as fine grained mineral sands. It is quite a different matter to extend the same principles and reasoning to poorly sorted sediments in the continental placer environment containing high-value minerals such as gold, platinum and diamonds.

It is particularly difficult to do so when the main values are contained in narrow paystreaks, gutters and potholes and where interpretation is made more difficult by the presence of large boulders. These boulders may be wrongly assumed to be part of the bedrock whereas, in fact, they lie upon bedrock and may conceal the presence of concentrations of valuable minerals around their bases. The result is an underestimation of both grade and volumes. Geostatistical methods may take into account the "nugget" effect when evaluating such deposits but even so they are subject to considerable error if the size of the samples taken fails to reflect adequately the geology of the deposit and the geometry of its surroundings.

Another factor is the effect on grade estimations of one particle of gold valued at 1 cent, ie. 1 mg Au with gold valued at $10 / gram. If such a particle is added or lost from the above pit sample, the grade of the

sample is raised or lowered by only $.01/m^3. Added to or subtracted from the borehole sample the grade differential becomes ± $1.27. Clearly, if the misplaced particle was valued at $1 instead of $.01, the resulting sample error would be scaled up or down by a factor of 100. In most cases, a reliable placer valuation will rely upon sample sizes and spacings that have been proven by adequate test work and experience of the engineer to be appropriate to the particular circumstances.

Faulty analytical procedures: The first consideration in sampling is to select suitable equipment and develop techniques for taking the samples that are is keeping with both the environment and the specific purposes of the investigation. This requirement applies equally to analysis as to the physical taking of the samples in that the analytical methods should not run contrary to the methods that will finally be used for recovery on a commercial scale. For example, some of the values may be locked up in particles of non-valuable materials that may not be recoverable by gravity methods. The use of fire assay or other instrumental analysis used as standard procedures in hard-rock ores, would give totally misleading results when used on placer ores. For similar reasons, placer samples should never be split for assay but the whole sample run as one unit to give more accurate results for the deposit.

Salting: A deposit is said to have been salted when drill holes are deliberately sited in more favorable locations, when quantities of the valuable mineral are introduced into samples after they have been taken, or when false measurements are substituted for true ones. It must always be assumed that salting will be attempted and precautionary measures including control samples in which blank samples are given normal code numbers and submitted along with other samples for analysis and statistical checks are made of actual samples for accuracy and symmetry.

Carelessness: This includes such faults as undue haste in preparing and dressing the sample area, accidental contamination, using approximate measurements instead of true ones to facilitate calculations and confusing sample labels. If sampling methods ore not standardized and rigorously followed, some if not all of these faults will inevitably occur.

Inaccurate measurements: Inaccurate marking out or leveling measured distances results in errors in calculating areas and volumes of influence and in geologic interpretation.

Drilling into a hard bottom: Values are lost when drilling into a bedrock unless it can be cleaned up by hand. Auger drilling onto a hard bedrock will always result in a loss of sample and in general an underestimation of values.

Use of drilling correction factors: The use of arbitrary corrections should be avoided by using systems that do not require corrections. Where necessary, the factors should be determined by careful studies under controlled conditions.

Inaccurate logging: Failure to recognize and delineate payable horizons and basement rocks may result in excessive dilution, conversely, in loss of mineral during both sampling and subsequent mining.

Unsuitable drilling equipment: Representative samples are unlikely to be taken if unsuitable gear is used. The type of hand auger suitable for sampling fine-grained mineral sands is quite unsuitable for sampling deposits containing large boulders.

Uneven distribution of values: Where the geometry of the bedrock is such that rich accumulations settle in potholes and other natural traps, the sampling pattern must be very close to avoid major errors. Such deposits tend to be undervalued unless very large bulk samples are taken.

Not allowing for dilution and batters: No placer is mined entirely within its payable limits. Batters must be allowed for along the sides of the excavation and a significant level of dilution results from mining along the boundaries between the gravels and barren overburden and from excavating into the bedrock. The amount of overburden to be allowed for depends upon the stratigraphic contours. A minimum of 10 inches above and below the gravels is normally included in the material excavated for treatment except where the bedrock is too hard to be ripped.

Ignorance: Probable more errors result from ignorance than from all of the other causes put together. All personnel should be fully trained in their respective duties and should not only understand how to perform the tasks assigned to them, but also why they should be done in a particular way.

Errors in continental placer sampling

These are due mainly to:

Splitting samples containing particles of widely different sizes. Such samples should be dressed as a whole.

Cutting irregular shaped pits and channels. All such samples are liable to error if the pit walls are not regular and vertical or is the channels are not regular in depth and section. The best method of sampling is to treat all of the material removed from a geometrically true section having vertical walls.

Errors in transitional placer sampling

These relate to grain counting and unnoticed changes in lithology.

Grain counting: This is an important application in minerals laboratory for spot testing and instrumental analysis. Errors are sue to the extremely small sample size used for observation, errors due to dividing the sample and screening the portions for analysis and errors from faulty identification of individual minerals.

Failure to recognize changes in lithology: Errors arise from the failure to recognize and evaluate changes in the lithology of individual strata during drilling, failure to recognize grain coatings or textural and other associations affecting marketability. Valuations may fail because operators expect similar responses to those of placers in other districts and have neglected to carry out pilot-scale investigations before designing the plant.

Errors in marine placer sampling

Specific errors may be due to :

Loss of heavy minerals during decantation of the sample, especially during bad weather conditions.

Irregular sample recovery due to the methods used: there is often a tendency for heavy minerals to be under-represented in the overlying sediments and for heavy minerals in the lower sediments to be over represented because of carry down and sorting in the column.

* Alluvial Mining, Eoin Macdonald, 1983, pages 227-233.

5-1

March 30, 1989 - Afternoon Session

Gold Losses at Klondike Placer Mines Gold Recovery Project

by Randy Clarkson P. Eng., Project Manager, New Era Engineering Corporation

Summary

Placer mining has made significant contributions to both the history and lifestyle of the Yukon and continues to provide a stable nongovernmental economic force. In 1988 there were approximately 185 active placer mining operations with a combined reported contribution in excess of $65 million to the Yukon's small resource based economy. Placer gold recovery at many of these operations is not optimized due to a lack of access to current technology, training, and testing facilities.

The objectives of the first phase of the Gold Recovery Project are to: collect representative tailings samples; recover the contained gold particles; evaluate and recommend improved recovery technology; and provide assistance in the selection and start up of recommended technology. The Klondike Placer Miners Association's (KPMA) Gold Recovery Project is unique because it is an industry initiated process research and development project which is under the direct control of the Yukon based placer industry.

Current gold recovery technology is almost exclusively confined to sluice boxes. A limited number of operations additionally employ feeders and screens. Sluice boxes are very simple, reliable, inexpensive, and yield very high concentration ratios. Many factors contribute to improved recoveries with a sluice box including:

a) controlled feed rates at less than 8 loose cubic yards/hr per foot of sluice width;

b) screened pay gravels to at least 3/4 inch;

c) adequate washing and liberation of free gold particles;

d) water ratio of 17 Imperial gallons/minute per loose cubic yard of pay gravels/hr;

e) use of both expanded metal and Hungarian riffles in every sluice run;

f) utilization of a slick plate section before a riffle section to allow gold segregation in the slurry;

g) even feeding through automation or strict manual control of loading equipment;

h) sluice box gradients of between 1.5 and 2 inches/foot; and

i) frequent removal of sluice box concentrates for upgrading.

Sluice boxes can recover up to 95 percent of gold particles as fine as 150 mesh provided that the previous conditions are realized. Sluice boxes may not be adequate for placer deposits containing very fine pay gravels with abundant clays and fine silts, a high proportion of high density minerals, or extremely fine (-150 mesh) or flattened gold particles. For a White Channel deposit, an oscillating sluice box kept riffles from packing and provided reasonable gold recovery.

Sampling design is critical for placer gold testwork because test results can be distorted by the 'nugget' effect. The uneven distribution of gold particles in a placer deposit often produces large random sampling errors. A sluice tailings stream represents one of the easiest sampling locations, provided the samples are caught in the air.

Sluice tailings samples were collected from six operating Yukon mines at regular intervals with hand held buckets, sample cutters, or a large steel box depending on the coarseness of the discharge (fig. G-1). The samples which represented between 1.5 and 7 loose cubic yards, were screened at 16 mesh and shipped to a tabling facility.

At each site, a large number of sample increments were alternately stored in two containers as interpenetrating samples over a period of 2 to 4 days. Comparison of these two samples indicated relative standard deviations (coefficient of variation) as low as 8 percent. These errors soared 56 percent (nugget effect) for two sites with high losses, when a limited number of gold particles as coarse as 14 mesh were found in their tailings.

Figure G-1: Artistic sketch illustrating sampling of screened gravels. Klondike gold recovery project.

To evaluate gold recovery efficiency, the collection of head samples is an mpractical task as well as being of dubious value. At each operation the sluice boxes were cleaned directly before and after the sampling period. A calculated head grade was determined by adding the gold recovered in the sluice box and the gold lost to the tailings. The placer gold data recorded over the 2 to 4 day sampling periods represent only a snapshot of a total deposit's characteristics.

Due to the difficulty in evaluating the losses of sluices, some operators and researchers have based their conclusions on errors in logic. These myths must be dispelled before the industry can progress.

The field sampling program was very expensive considering that only six sites were sampled. However, the recoverable losses which have been identified at two of the sites (C and F) would pay for the ntire program after 1 year of operation. This research has a much greater benefit o the entire placer industry as other miners incorporate these recommendations into their sluicing systems.

Overall Losses

Overall losses ranged between 0.0006 ounces per loose cubic yard (oz/Lyd³) or $7.57 per operating hour at site A to 0.0021 oz/Lyd³ or $130 per operating hour at sites C and F. As much as 66 percent of these losses (or $140,000 per season) could be recovered chiefly by reducing feed rates to 8 Lyd³/hr per foot of sluice width and by screening the pay gravels to 3/4 inch.

At all of the sites the majority of the gold values lost to the tailings were coarser than 48 mesh. Tabling testwork detected very little gold finer than 150 mesh in the tailings. With the exception of site E, gold lost to the tailings had the same shape (Corey Shape Factor) as the gold recovered by the sluice boxes.

Estimated improvements to recoveries were based on comparisons between sites described by Poling (1986), and Zamyatin and others (1975) published data. There is a very limited amount of reliable data regarding the recovery of jigs and other gravity concentration devices.

Effect of Screening on Gold Recovery

The tailings sampling program clearly demonstrates the effect of screening on gold recovery with conventional sluice boxes. Figure G-2 illustrates the recovery of sites A, D, E, and F versus gold particle size. Sites A, D, and E had evenly fed sluice boxes with optimal feed rates. However, site A was screened to 3/8 inch and recovered 95 percent of gold as fine as 150 mesh. Site D was screened to 2.5 inches and recovered gold coarser than 65 mesh fairly well.

Site E, an unscreened single nonsluice boxes recovered very little of the gold coarser than 65 mesh but still recovered more than site F's triple run sluice box. Even though site F's overall feed rate was nearly optimal, its stationary punch plate distributor was a very inefficient screen (40 percent) and directed the majority of the -3/4 inch gravels down the heavily overloaded center run, resulting in heavy losses (12 percent overall).

These results agree with published data from the Soviet Union (Zamyatin and others, 1975). Figure G-3 illustrates the effect of reducing the top size of pay gravels from 5/8 inches to 5/32 inches on the recovery of a shallow fill (expanded metal riffles) sluice box. Dramatic recovery improvements can result when the pay gravels are screened. Riffle packing can occur due to a high proportion of either fine silts and clays or heavy minerals in the pay gravels. These two conditions limit the application of conventional sluice box recovery systems.

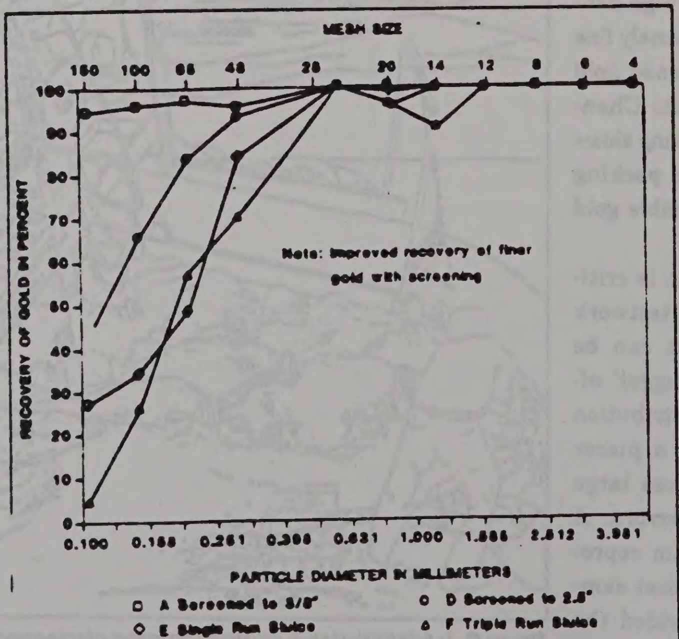


Figure G-2: Gold recovery vs. particle diameter. Yukon placer recovery study.

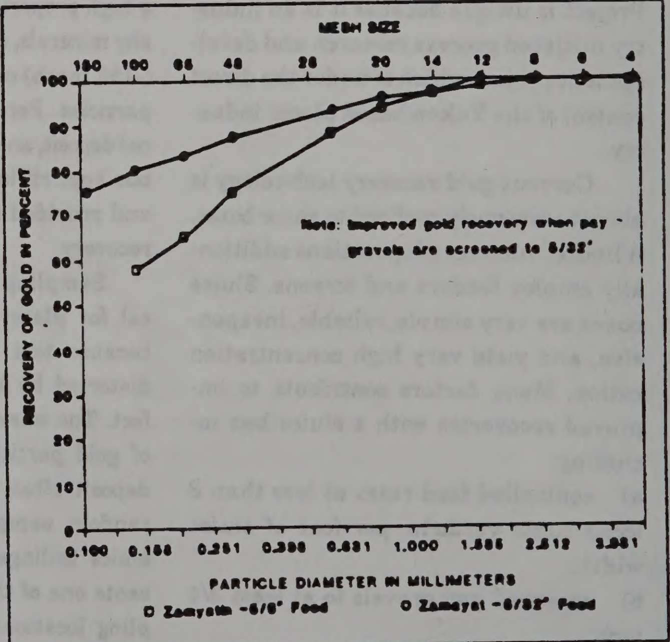


Figure G-3: Gold recovery vs. particle diameter. Soviet placer recovery study.

Effect of Riffle Type on Gold Recovery

Figure G-4 illustrates the effect of riffle type on gold recovery. Data from site E, F, and Zamyatin and others (1975) are represented. Site E's sluice box consisted of a top section of expanded metal riffles directly under 3/4 inch punch plate and its bottom section was lined with Hungarian riffles. The expanded metal riffles were much more efficient at recovering gold finer than 1 mm (14 mesh); however they were unable to retain much of the coarser gold. The coarse gold which passed through the expanded metal riffles was caught by the Hungarian riffles.

These conclusions are mirrored in site F's side runs (containing expanded metal) and center run (Hungarian riffles), although other factors such as overloading are additionally responsible for the center run's high losses. Many researchers have recommended the use of Hungarian riffles as a 'Nugget Trap'; however, most did not realize just how small a nugget could be. The coarse gold losses with expanded metal are very dramatic as the particle size increases.

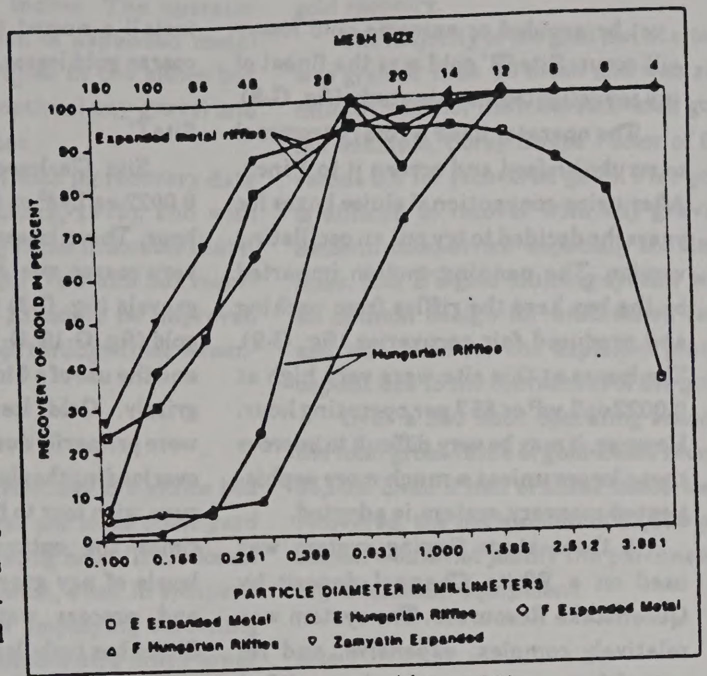


Figure G-4: Effect of riffle type on gold recovery.

Individual Site Evaluations

Site A

Site A's exceptional gold recovery is illustrated in figure G-5. Site A's gold losses were the lowest recorded at $7.57 per operating hour or 0.00055 oz/Lyd3. This was due to the following good recovery practices: removal of all +3/16 inch feed gravels; feed regulation by a hopper and vibrating screen; and use of efficient expanded metal riffles. Site A's water use is very low compared to recommended values but this does not appear to have affected gold recovery.

Distribution of gold values in site 'A' (fig. G-6), shows that most gold is fine sized and that most of the lost gold values are between 48 and 28 mesh in size. An unscreened sluicing system with poorer recoveries of fine gold would lead to extreme losses of gold at this site. Additional losses at this site were due to the loss of fine gravels which adhered to boulders rejected by the dry grizzly. The sluice contained only expanded metal riffles; therefore there may also be losses of gold coarser than 1 mm.

Site B

Site B's pay gravels are one of the exceptions to the applications of conventional sluice boxes. The White Channel gravels located at this site contained a high percentage of clays and fine silts which are difficult to wash and tend to pack the riffles (fig. G-7). Riffle packing

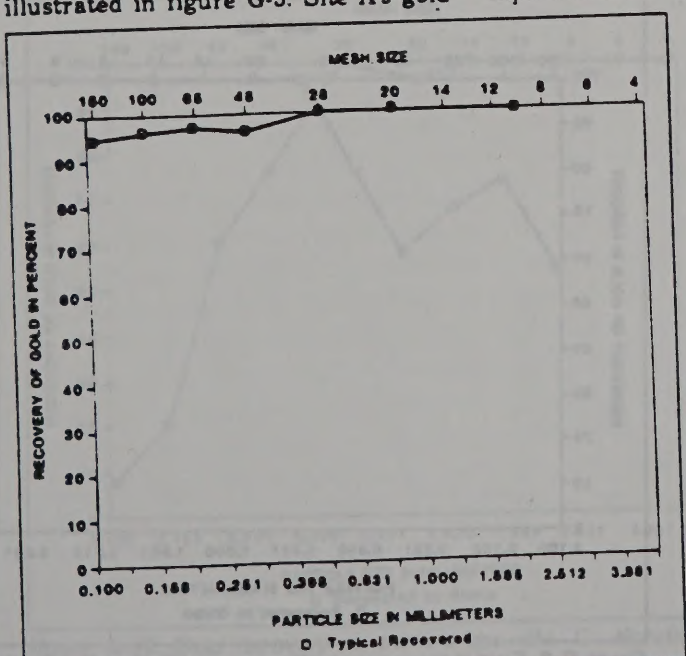


Figure G-5: Gold recovery vs. particle size, site 'A', Klondike District, Yukon.

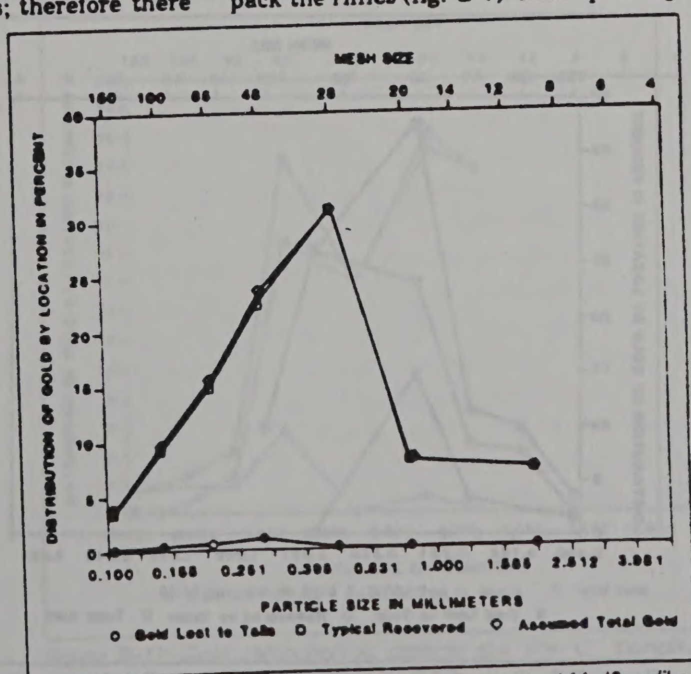


Figure G-6: Gold distribution vs. particle size, site 'A', Klondike District, Yukon.

ust be avoided or extreme gold losses will occur. Site 'B' gold was the finest of ny investigated in this study (fig. G-8).

The operator used a rotary trommel to scrub the feed and screen it to 1 inch. After using conventional sluice boxes for years, he decided to try out an oscillating version. The panning motion imparted by the box kept the riffles from packing and produced fair recoveries (fig. G-9). The losses at this site were very high at 0.0022 oz/Lyd³ or \$57 per operating hour. However, it may be very difficult to improve these losses unless a much more sophisticated recovery system is adopted.

A three stage jigging system was used on a White Channel deposit by Queenstake Resources. The system was relatively complex, expensive, and required frequent adjustment by qualified personnel. A very rough comparison with published jig recovery (Zamyatin and others, 1975) indicates that a jigging vstem would provide additional values f between \$10 to \$20 per hour. This is quivalent to additional revenues of \$10,000 per 1,000 hour operating season, not enough to justify the purchase and operating expenses of a jigging system without further full scale testwork. The oscillating sluice run used only expanded metal riffles. The operator intends to install a nugget trap to guard against coarse gold losses.

Site C

Site C's losses were very high at 0.0022 oz/Lyd³ or \$126.04 per operating hour. These losses occurred despite the very coarse size distribution of its pay gravels (fig. G-7) the coarseness of the gold (fig. G-10; G-11) and the use of a 6 inch grizzly. Gold losses were primarily due to overloading the sluice runs with four to five times the optimum levels of pay gravels and process water. Sluice box turbulence was worsened with the passage of minus 6 inch rocks. In addition the sluice run used hungarian riffles which are less efficient at recovering gold particles smaller than 1 mm.

Increased recoveries should be achieved by making the sluice box four times wider. Process water volumes would also have to be increased to move coarse rocks down the sluice run. Improved recoveries should also result from the use of expanded metal riffles over Nomad matting instead of Hungarian riffles. Hungarian riffles should be used in the last section of the sluice box (at least 8 feet) to trap gold particles coarser than 1 mm.

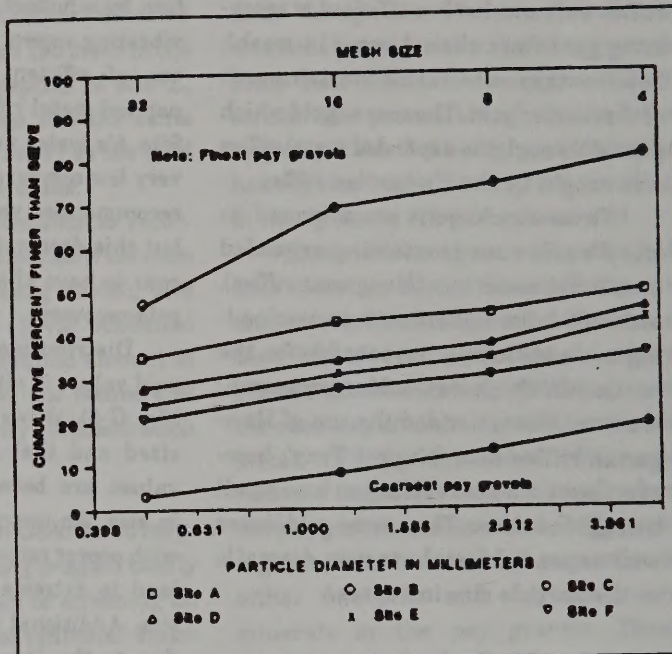


Figure G-7: Feed gravel size distribution, Klondike District, Yukon.

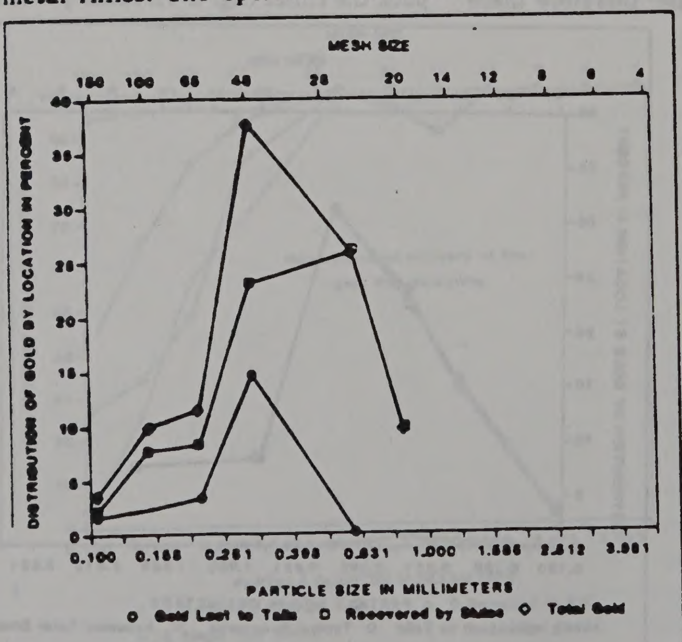


Figure G-8: Gold distribution vs. particle size, site 'B', Klondike District, Yukon.

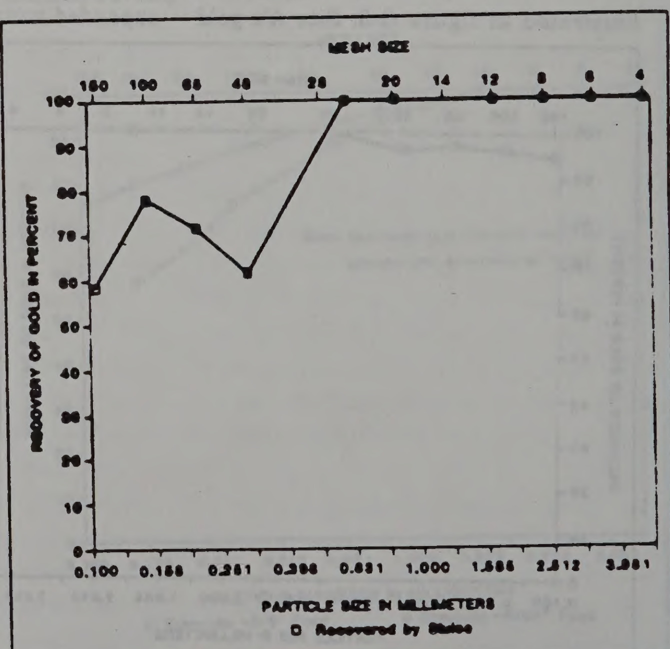


Figure G-9: Gold recovery vs. partile size, site 'B', Klondike District, Yukon.

Expanded metal riffles cannot tolerate the wear resulting from the passage of coarse rock. Therefore, the pay gravels should be screened to 3/4 inches. Screening could be accomplished by suspending metal punch plate above the expanded metal riffles (similar to site E), or preferably by utilizing a vibratory screen ahead of the sluice. The vibrating screen would also promote better washing and reduce water requirements dramatically.

Comparison with site F's side runs indicates that a screened sluicing system should recover half of the lost gold or $62.71 per hour ($50,200 per operating season). A further comparison with site E's recovery indicates that an unscreened system with suspended punch plate would recover $26.37 per hour or $21,100 per season of lost gold values.

Site D

Gold losses at site D amounted to only 0.0010 oz/Lyd³ or $34.49 per operating hour despite its relatively fine pay gravels (fig. G-7) and abundance of heavy minerals. This operator attempted to minimize riffle packing problems by shutting down his sluice box and hand raking the riffles every 2 hours and it appears to have worked. A Derocker system provided feed regulation and screening to 2.5 inches. The operator used a combination of expanded metal and Hungarian riffles in the sluice box which was fed at optimal pay gravel and process water rates.

A comparison with jig recovery data (Zamyatin and others, 1975), and with site C's oscillating sluice indicates that it is doubtful that site D's sluice box recovery (fig. G-12, G-13) could be improved significantly except through finer screening.

Site E

Gold losses from site E's sluice box total 0.0011 ounces per loose cubic yard or $22.24 per operating hour. These losses are among the lowest, even in comparison to operations employing screening and feeding equipment and much lower volumes of wash water.

Site E's sluicing equipment recovered much less of the fine gold (-48 mesh) than operations employing screening equipment (fig. G-14); however, there was almost no fine gold in the original pay gravels (fig. G-15). Feed rates, slope, and riffle layout were optimal, but the water ratio was three times higher than recommended. The high water levels which were required to push coarse rocks down the sluice run did not appear to affect gold recovery.

The majority of the gold lost at site E was greater than 48 mesh and was significantly flatter than the recovered gold (for example, Corey Shape Factor of 0.2 versus 0.4 for recovered gold). Flat gold is difficult to recover with any gravity concentration device--especially the finer sizes. Site E's gold sluicing system was an optimal design for unscreened feed and was ideal for the exploited placer deposit due to the coarseness of the gold.

Over a 250 hour operating season, the total gross value of gold losses is only $5,500. Even if half of these losses were recovered, the net amount of $2,300 per season would not justify the purchase of any additional equipment.

Site F

Gold losses from site F amount to 0.0021 oz/Lyd³ or $132 per hour. Gold as coarse as 14 mesh was recovered in the center and side runs of this triple-run sluice (figs. G-16 and G-17). Recoveries in the dump box and side runs were poor compared to other sites. The center run's recovery system was even worse and lost almost all of the gold finer than 28 mesh.

The pay gravels were pushed onto the side of the dump box, which was covered with 3/4 inch punch plate. The

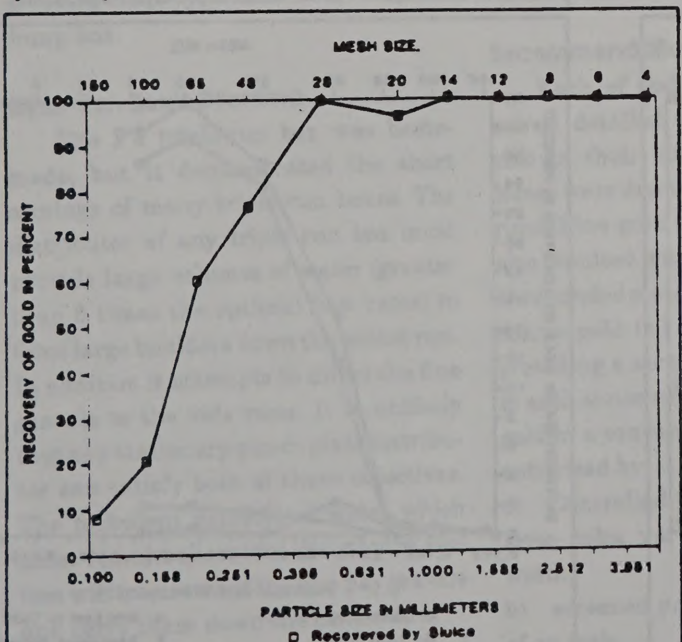


Figure G-10: Gold recovery vs. particle size, site 'C', Klondike District, Yukon.

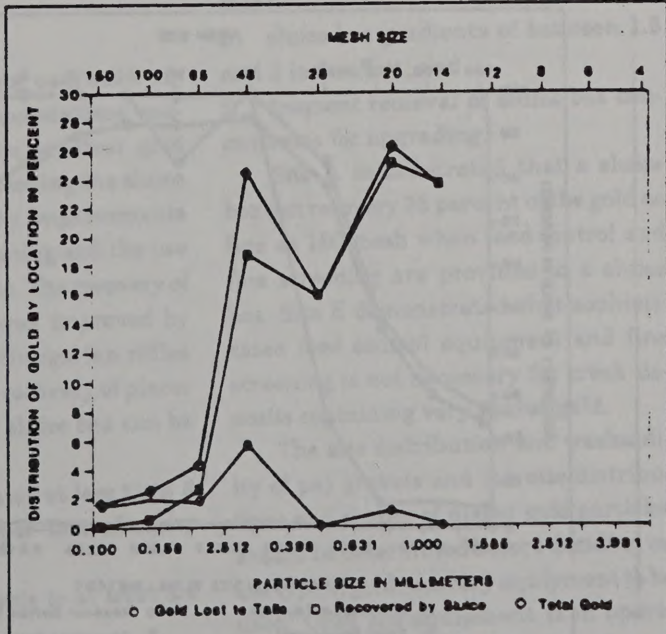


Figure G-11: Gold distribution vs. particle size, site 'C', Klondike District, Yukon.

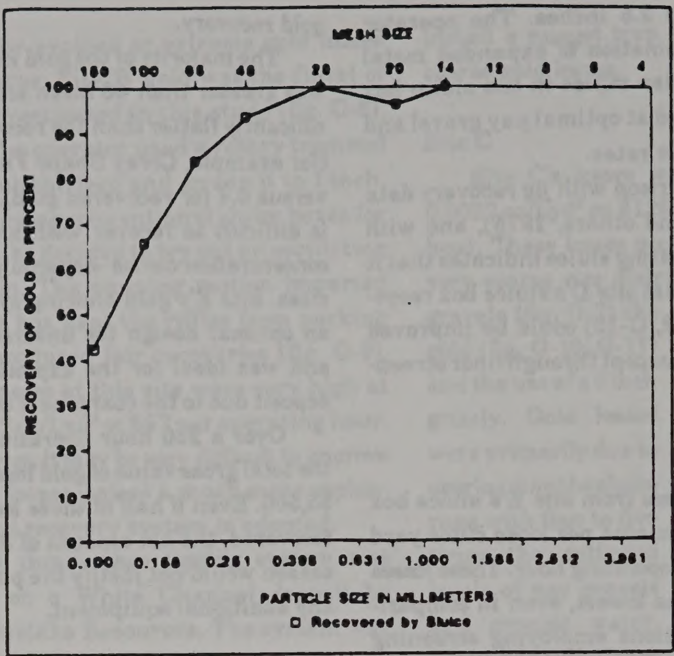


Figure G-12: Gold recovery vs. particle size, site 'D', Klondike District, Yukon.

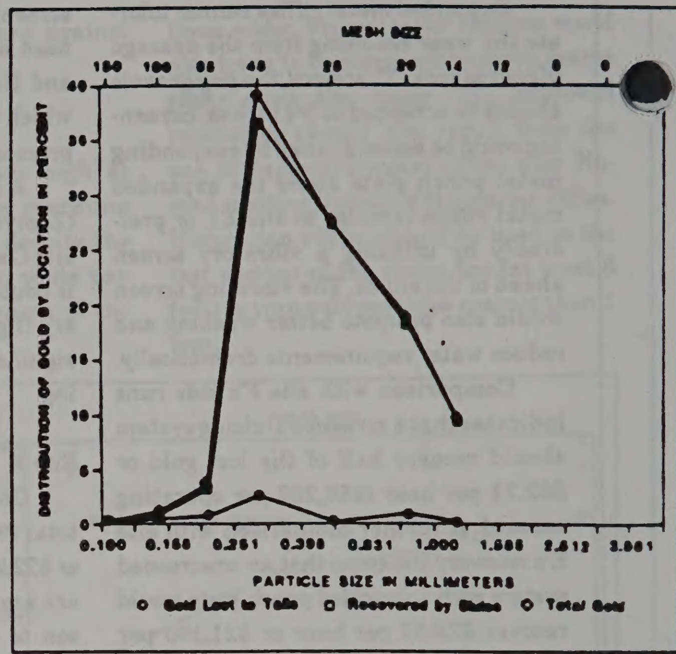


Figure G-13: Gold distribution vs. particle size, site 'D', Klondike District, Yukon.

gravels were forced down the box by a stationary monitor to the distributor. This 3/4 inch punch plate section directed the underflow to two side runs, while the overflow continued on to the center run. An undercurrent sluice run was located under the final section of the center run. Expanded metal riffles were used in the dump box and side runs, while Hungarian riffles were used in the center run. All of the riffles were underlain with backed Nomad matting.

These high losses were primarily due to the extremely poor screening efficiency of the distributor section. Despite the large area of its punch plate (46 ft²), less than 40 percent of the fine gravels were distributed to the side runs. The poor screening efficiency (40 percent) of the distributor resulted in a feed rate which was four times the optimal to the center run and 1/4 the optimal to the side runs. The center run was overloaded and had to pass large boulders and high volumes of water which created excessive turbulence. In addition, the bulldozer spent only 5 percent of its cycle feeding the dump box and this resulted in surging feed rates.

If the recovery of the center run could be increased to that of the side runs, an additional $87 per operating hour or $140,000 per operating season of

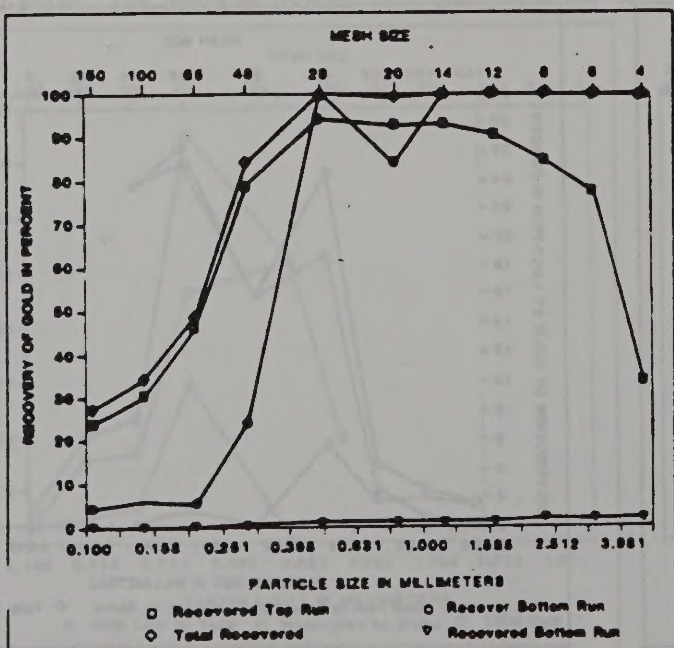


Figure G-14: Gold recovery vs. particle size, site 'E', Klondike District, Yukon.

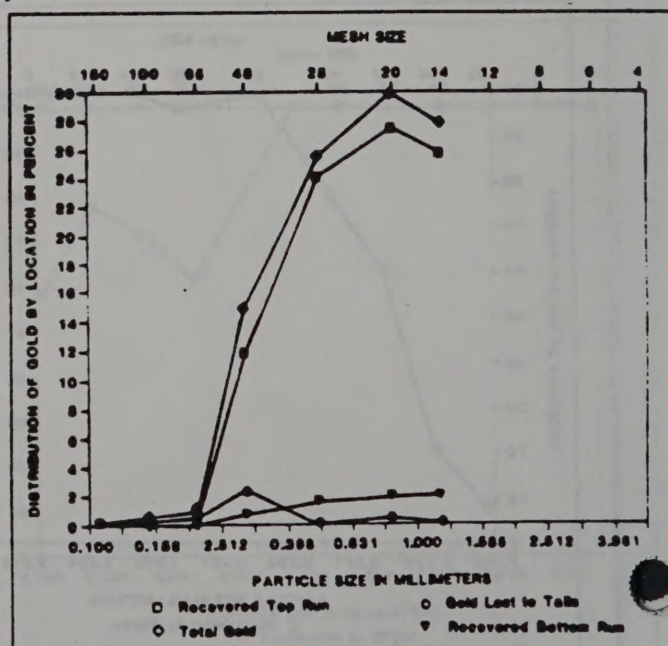


Figure G-15: Gold distribution vs. particle size, site 'E', Klondike District, Yukon.

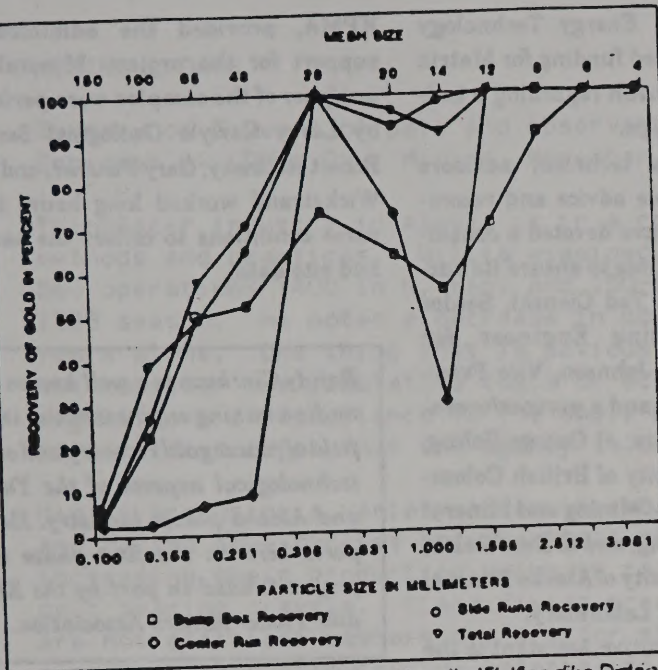


Figure G-16: Gold recovery vs. particle size, site 'F', Klondike District, Yukon.

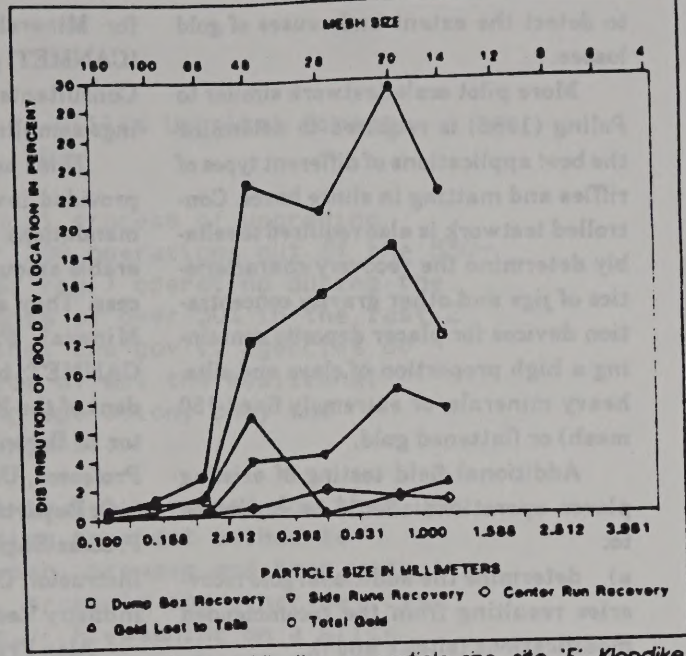


Figure G-17: Gold distribution vs. particle size, site 'F', Klondike District, Yukon.

gold would be recovered. This could be accomplished by screening the pay gravels prior to sluicing with a rotary trommel or Derocker-type assembly. The side runs also lost gold as coarse as 14 mesh. These losses could be reduced by including a section of Hungarian riffles at the end of the sluice runs. Gold recovery could be improved further by controlling feed surges with a manned monitor and devoting more cycle time to feeding the dump box.

Triple-Run Box's Dilemma

Site F's triple-run box was homemade, but it demonstrated the short comings of many triple run boxes. The distributor of any triple run box must provide large volumes of water (greater than 5 times the optimal flow rates) to force large boulders down the center run. In addition it attempts to direct the fine gravels to the side runs. It is unlikely that any stationary punch plate distributor can satisfy both of these objectives. The turbulent volumes of water which must remain above the distributor section will trap some of the fine pay gravels and carry them down the center run.

The center run is normally lined with Hungarian riffles and is extremely turbulent due to the passage of boulders and high water volumes. Therefore it is unlikely that reasonable gold recoveries will result in this type of sluicing environment. Many operators have added an undercurrent to improve the screening action and recovery efficiency of their center runs. However, the recovery characteristic of site F's center run did not appear to be improved by the inclusion of its undercurrent sluice.

Recommendations

Each of the six site operators received detailed recommendations specific to their sites. The greatest gold losses were due to overfeeding the sluice runs. Fine gold recovery improvements also resulted with screening and the use of expanded metal riffles. The recovery of coarse gold (+1 mm) was improved by providing a section of Hungarian riffles in each sluice run. The recovery of placer gold in a conventional sluice box can be optimized by:

a) Controlled feed rates at less than 8 loose cubic yards/hr per foot of sluice width;

b) screened pay gravels to at least 3/4 of an inch;

c) adequate washing and liberation of free gold particles;

d) water ratio of 17 Imperial gallons/minute per loose cubic yard of pay gravels/hr;

e) use of both expanded metal and Hungarian riffles in every sluice run;

f) utilization of a slick plate section before a riffle section to allow gold segregation in the slurry;

g) even feeding through automation or strict manual control of loading equipment;

h) sluice box gradients of between 1.5 and 2 inches/foot; and

i) frequent removal of sluice box concentrates for upgrading.

Site A demonstrated that a sluice box can recovery 95 percent of the gold as fine as 150 mesh when feed control and fine screening are provided to a sluice box. Site E demonstrated that sophisticated feed control equipment and fine screening is not necessary for creek deposits containing very coarse gold.

The size distribution and washability of pay gravels and the size distributions and shapes of placer gold particles should be determined before deciding on the type of gold recovery equipment to be used. Once the equipment is in operation, periodic tests should be conducted

to detect the extent and causes of gold losses.

More pilot scale testwork similar to Poling (1986) is required to determine the best applications of different types of riffles and matting in sluice boxes. Controlled testwork is also required to reliably determine the recovery characteristics of jigs and other gravity concentration devices for placer deposits containing a high proportion of clays and silts, heavy minerals, or extremely fine (-150 mesh) or flattened gold.

Additional field testing of existing placer operations should be conducted to:

a) determine the additional gold recoveries resulting from the recommended modifications (sites c and f);

b) optimize the performance of a three run box;

c) expand the knowledge of gold recovery at a greater variety of deposit types and recovery equipment.

The use of nuclear tracers in field testwork should be investigated. Nuclear tracers may be able to reduce sampling costs; increase the number of sites investigated; and provide quantitative data for operations where collecting samples is very difficult or dangerous due to the passage of coarse rocks.

Acknowledgments

The National Research Council's Industrial Research Assistance Program (IRAP) cost shared the following: salaries and administration costs for the project manager and his field assistants; sampling consulting services (Jan Merks of Matrix Consultants); and laboratory services (Bud Kenzie of Midas Concentrating Company). The Canada/Yukon Economic Development Agreement provided an additional $60,600 towards costs incurred during the field sampling and analysis program. The Canada Center for Minerals and Energy Technology (CANMET) provided funding for Matrix Consultants to consult regarding a tailings sampling design.

This project's technical advisors provided invaluable advice and recommendations and have devoted a considerable amount of time to ensure its success. They are: a) Tad Cienski, Senior Mineral Processing Engineer for CANMET; b) Ron Johnson, Vice President of the KPMA and a partner/operator of Beron Placers; c) George Poling, Professor, University of British Columbia's Department of Mining and Mineral Process Engineering; and d) Dan Walsh, Instructor, University of Alaska Mineral Industry Research Laboratory.

Alan Fry, Executive Assistant to the KPMA, provided the administrative support for the project. Mineralogical analyses of the samples were performed by Larry Carlyle Geological Services. Robert Rebinsky, Gary Pletcher, and Merle Wickstrand worked long hours in adverse conditions to collect the samples and site data.

Randy Clarkson is a well known Canadian mining engineer active in the field of placer gold recovery and other technological aspects of the Yukon and Alaska placer industry. He reports here on the first phase of a project funded in part by the Klondike Placer Miners Association.

Editors note - a more complete summary of Clarkson's research including a historical review of placer mining in Yukon is available from Klondike Placer Miners Association, P.O. Box 4927, Whitehorse, Yukon Territory, Canada Y1A-3V1.

References

Poling, G.W., Hamilton, J.F., 1986, Fine gold recovery of selected sluice box configurations, University of British Columbia: prepared for the Canada/Yukon Economic Development Agreement, 200 Range Road, Whitehorse, Yukon Territory, Canada.

Zamyatin, O.V., Lopatin, A.G., and others, 1975, The concentration of auriferous sands and conglomerates: published by Nedra Press Moscow. Translation available from the Geological Society of Canada, Ottawa, Ontario.

MINING UPDATE

Summary of 5 years of data and observations by Willis Umholtz, Mining Engineer, Ak. DNR, Div. Mining, November 21, 1989.

The placer industry in Alaska is in a continual process of upgrading methods and practices. Willis examined over 200 operations out of the 550-560 operations (400 in N. reg. and 150-160 S. reg.) operating during the 1989 season. He noted a decrease in pollutants of over 50% in the last 2 years alone. One thing that is obvious is that the gov't. agencies do not consider the cumulative costs or economics of all the additional regulations or compliance to the whole mining operation, only the particular portion that the agency is in charge of.

Miners are using a variety of equipment that is not intended for that use and having some success but are still not using the best method to increasing their production which is to prewash, prewash and then wash the gold bearing gravels. Classifiying screens, trommels and other devices are not efficient washers but are for sizing. Prewashing on a slick plate and then allowing the material to flood down the sluice box will give better recovery in a riffle system than trying to make a screen prewash and classify the gravels.

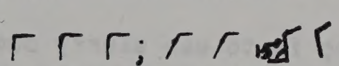
Losses in sluice boxes of same dimensions with different riffles are:

expanded metal screen	10% with most in +10 mesh fraction
dredge riffle	4% with most in the - 60 to 100 mesh
hungarian riffle	4% with most in the - 60 to 100 mesh

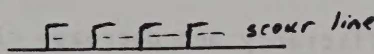
If there are only hungarian riffles in the box, there is a 14% loss of gold regardless of the size of gold or slope of the box.

Cocoa and nomad type matting give equal recovery results while astroturf loses more gold than the others.

Hungarian riffles with a tip up angle of 7-10° do not work as well in recovery as those that are horizontal in the flow.

ok:  no: 

The scour line behind the riffle should not exceed one half the height of

the riffle: flow --> 

Hungarian riffles work best at the end of the sluice box and really should not be used on the head end where they create too much turbulence, compared to a thick (7/8 ") expanded metal riffle, and allow the gold to migrate down the sluice.

Optimum efficiencies for sluices appears to be ≈ 80% using a combination of expanded metal and hungarian riffles; if hungarian only, efficiency is 70%.

RULE OF THUMB TO OPTIMIZE RECOVERY

Maximize: water usage: 4 lb. H_2O : 1 lb. feed gravels

Grade of box: 2 - 2.5 inches per foot

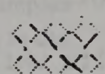
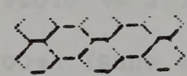
These values will optimize recovery while allowing the smallest of effects from feed slugging or starving the sluice box.

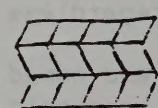
Average grade values for sluices are too flat and better recovery will be found around 2 to 2.5" per foot. The value of sluice boxes of 1.5 to 1.75 inches per foot came from the bucket line dredge sluices and is not valid for stationary plants.

To adequately sample the losses to the tailings, one must take 40-100, 16" pans of material per day at regular intervals (ie. 10 pans/ hour).

Better recovery is obtained on vibrating screen decks if the material is thoroughly slurried at least one foot to 18 inches ahead of the screen rather than trying to wash and screen at the same time. The faster and better the feed material is wetted and prewashed, the better screening and classification can occur and gold recovery will be increased, all else remaining the same. Better slurrying is obtained if the material is prewashed over plastic sheeting (polyethelene .75 to 1" thick) placed ahead of the screens or riffles rather, than in a hopper or trommel.

The opening size of expanded metal is not as critical as the thickness or shape of the opening. Best recovery obtained from 5/8 to 7/8 inch thick screen that is in an off-set pattern.

ok: flow: -->  or ^  flow

not --> flow 

Expanded metal riffles will recover 95% of the -20 to +100 mesh gold fed across the riffle.

The best mechanical system for prewashing is to use slurry pumps or hydraulic lifts. Even though lifts exceed the optimum water balance by double to 8:1, water to feed ratios.

One method to increase sluice efficiency and decrease clean up time is to build a "boil box" positioned 12" above the first riffle section. This is a box built into the bottom and across the full width of the sluice. The box is 4" wide by 4" deep on the head end of the sluice and 8" deep on the lower end with a water manifold along the bottom with spray openings aimed upward. The top of the box is fitted with punch plate with 1" holes. At one end of the trough or box a pipe drain is built in and capped. This is used to clean out the boil box of nuggets. The boil box may also serve as a sampling devise to be used to sample each 100 yards of material processed in the sluice box.

Another clean out devise is built into the bottom of the sluice about 6' below the boil box. This is a 4" wide box 4" deep set at a 45° angle across the width of the sluice. The lower end of the trap is fitted with a drain and pipe cap. The trap is covered with the carpet and riffles of the sluice and then during clean up, all the material caught in the riffles and on the carpet is washed out above the trap. Water is then used to wash the material into the trap and out the drain into a bucket.

AVERAGE COSTS (1985 $

Stripping:

hydraulic:	40-50 ¢ per yard
mechanical:	$1.50 - 3.00 per yard

HOW THOROUGH SHOULD A PLACER INVESTIGATION BE?

What is an adequate placer investigation?

In private practice, a mineral examination is simply a process of testing the successive links in a chain of observed facts. As soon as one link in the chain is found to be fatally weak - the examination is over.

In the case of some placer operations, the conditions observed on the ground will often tell the experienced mineral examiner that the proposed mining venture will result in certain failure.

For example, a property may have a rough, hard bedrock that outcrops in many places or the gravels may be very thin. This would indicate that a bucket line dredge would not be an applicable mining method. If numerous boulders are in evidence or if they are of consistently large size, the operation will be more costly than if the gravels are uniform in size and can be handled by conventional sized mining equipment. The cost of meeting surface reclamation and water quality standards may also prohibit a mining operation with low grade gravels and with limited reserves from being a profitable mine. What ever the weak link is, in private industry practice, the engineer's statement that the property merits no further consideration will usually end the examination.

But as government mineral examiners, we must go beyond this first fatal link determination; and for good reason. The government mineral examiner must expect to become part of an adversary proceeding as a result of their mineral examination. As an expert witness for the government, the mineral examiner must convince a hearings judge that not only an adequate job was done but a professional job as well. The examiner must show that all the pertinent facts were analyzed and their effects on the mining operation were considered and not just the obvious ones. To simply point out a fatal link is not enough. A substantial knowledge of the size and probable tenor of the deposit, its geologic setting, an intelligent consideration of mining methods and costs of the operation, including meeting water and reclamation standards, will enhance the value of the mineral examiner's testimony in an adverse situation.

How thorough should an examination be? The answer is what ever is necessary to support the testimony and conclusions of the mineral examiner when testifying as an expert witness. The finding of only one fatal link may not be enough when in a contest or adversary proceeding.

BUCKET VOLUME
5 - gallon plastic

INCHES	VOLUME
1	.0455
2	.0910
3	.1365
4	.1846
5	.2326
6	.2807
7	.3339
8	.3872
9	.4433
10	.4995
11	.5558
12	.6121
13	.6648

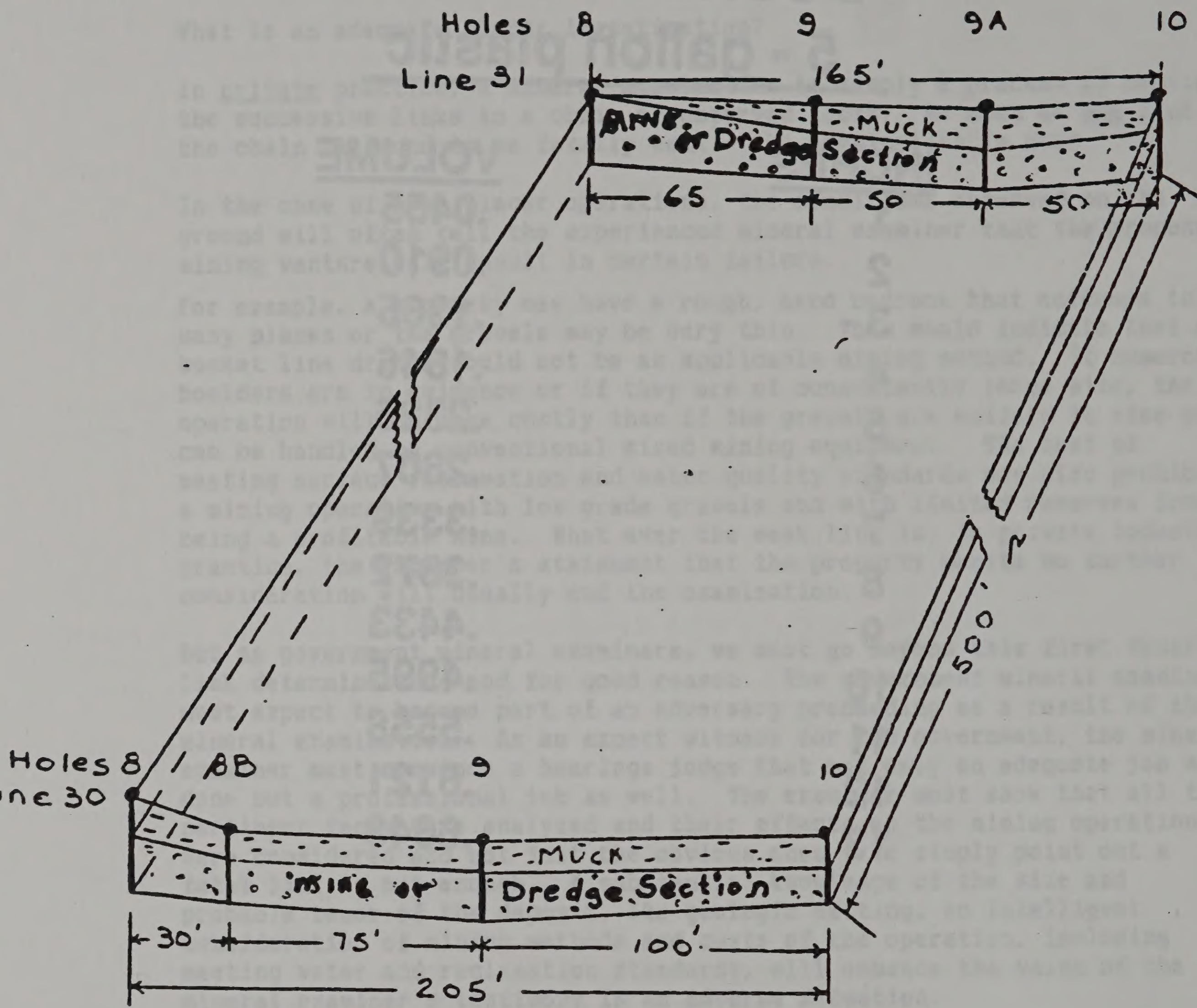


Diagram Of Placer

Block Estimate

Scale 1"=50'

BLOCK ESTIMATE

Line No.	Hole	Spacing	Depth Mk.	Depth DS	Value $/Cu. Yd.	Muck X Spacing	D.S. X Spacing	D.S.X Spacing X Val./Cu. Yd ($)
30	8		11	16	16.9	330	480	81
	8B	30	8	12	35.2	840	1260	444
	9	75	9	14	196.4	1575	2450	4812
	10	100	10	10	204.9	1000	1000	2049
Total		205	38	52	453.4	3745	5190	7386
Average End Area			9.5	13	113.4	1873	2595	$ 3693

31	8		0	19	77.9	0	1235	962
	9	65	8	12	64.5	920	1380	890
	9A	50	12	11	120.0	1200	1100	1320
	10	50	9	12	55.9	450	600	335
Total		165	29	54	318.3	2570	4315	3507
Average End Area			7.2	13.5	79.6	1285	2158	$ 1754

BLOCK VALUATION

Cubic Yards Muck

$$\begin{array}{r} 1873 \\ \underline{1285} \\ 3158 \end{array} \qquad \frac{3158 \times 500}{54} = 29,240 \longleftarrow$$

$\frac{1}{2}$ = 27 ft²/yd = 2 ends

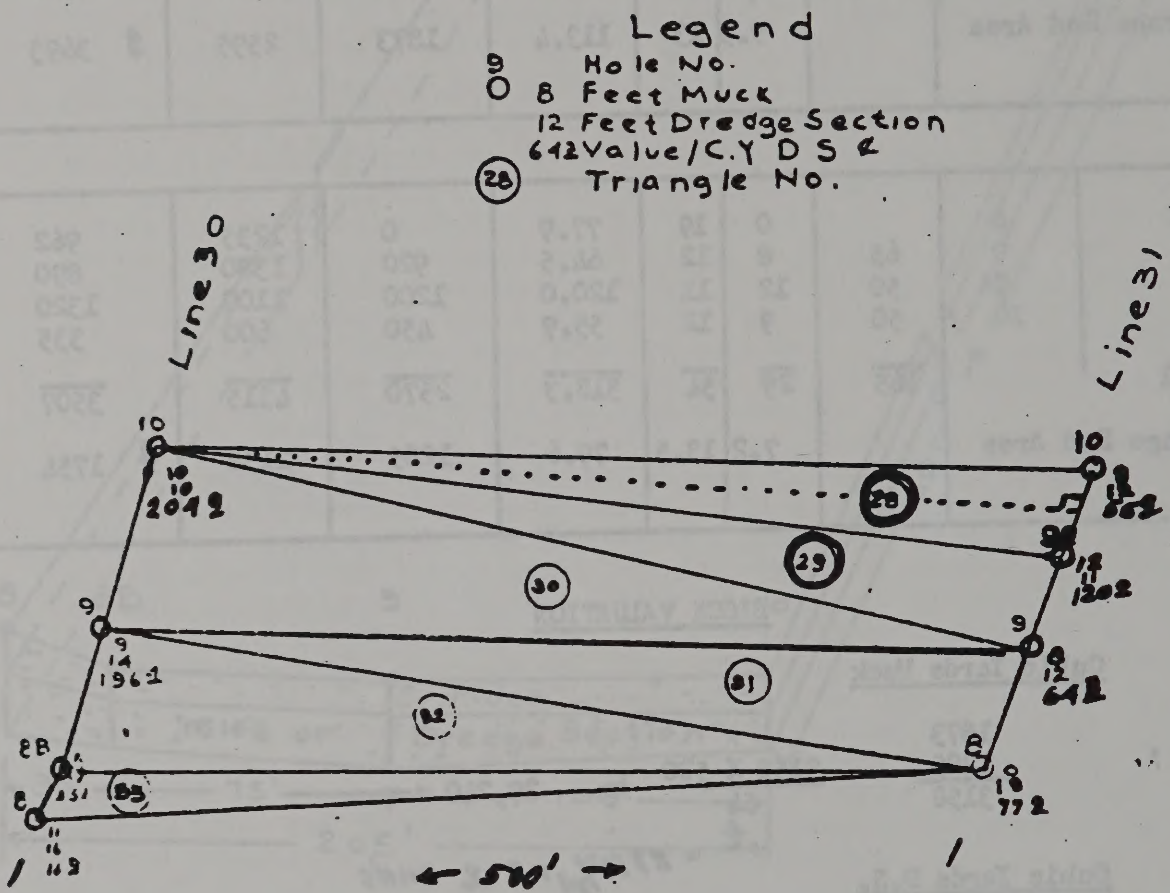
Cubic Yards D.S.

$$\begin{array}{r} 2595 \\ \underline{2158} \\ 4753 \end{array} \qquad \frac{4753 \times 500}{54} = 44,009 \longleftarrow$$

Ground Value $

$$\begin{array}{r} 3693 \\ \underline{1754} \\ 5447 \end{array} \qquad \frac{5447 \times 500}{54} = \$\,50,435 \longleftarrow$$

CH 21



Valuation Triangles
Scale 1" = 100'

TRIANGLE ESTIMATE

			a	b	c	b · c	a/3	b/3	Σb·c / Σb	I / 27	II / 27	III · IV
Tri.	Line	Hole	Depth Mk.	Depth D.S.	Val. C.Y. D.S. $	Val. X Depth D.S.	Ave Depth Mk.	Ave Depth D.S.	Val. C.Y. DS $	Factors/ Cu. Yd. Mk.	Factors/ Cu. Yd. D.S.	Sq. Ft. Value $
							I	II	III		IV	4
28	31	10	9	12	.559	6.72						
	31	9A	12	11	1.200	13.20						
	30	10	10	10	2.049	20.49						
			31	33		40.41	10.3	11.0	1.23	0.382	0.407	0.502
29	31	9A	12	11	1.200	13.20						
	31	9	8	12	0.643	7.70						
	30	10	10	10	2.049	20.49						
			30	33		41.39	10.0	11.0	1.25	0.370	0.407	0.510

TRIANGLE VALUATION

	1	2	1×2	3	1×3	4	1×4
Tri. No.	Area Sq. Ft.	Muck Factor	Cu. Yds. Muck	D. S. Factor	Cu. Yds. D. S.	Value Factor	Gross $ Value
28	11,800	0.382	4,507	0.407	4,803	0.502	5,924
29	13,150	0.370	4,866	0.407	5,352	0.510	6,706
30	24,950	0.333	8,308	0.443	11,053	0.688	17,166
31	15,750	0.207	3,260	0.557	8,772	0.616	9,702
32	21,000	0.207	4,347	0.557	11,697	0.570	11,970
33	5,850	0.233	1,363	0.582	3,405	0.288	1,685
Total	92,500		26,651		45,082		53,153

1-25

COMPARISON OF VALUATION METHODS

System	Area Sq. Ft.	Muck Cu. Yds.	D.S. Cu. Yds.	Gross $
Block	92,500	29,200	44,000	50,400
Diamond	92,500	27,855	44,280	51,396
Triangle	92,500	26,651	45,082	53,153
Average	92,500	27,902	44,454	51,649

C4 23.

Table 4
Mining Cost Comparison Summary +

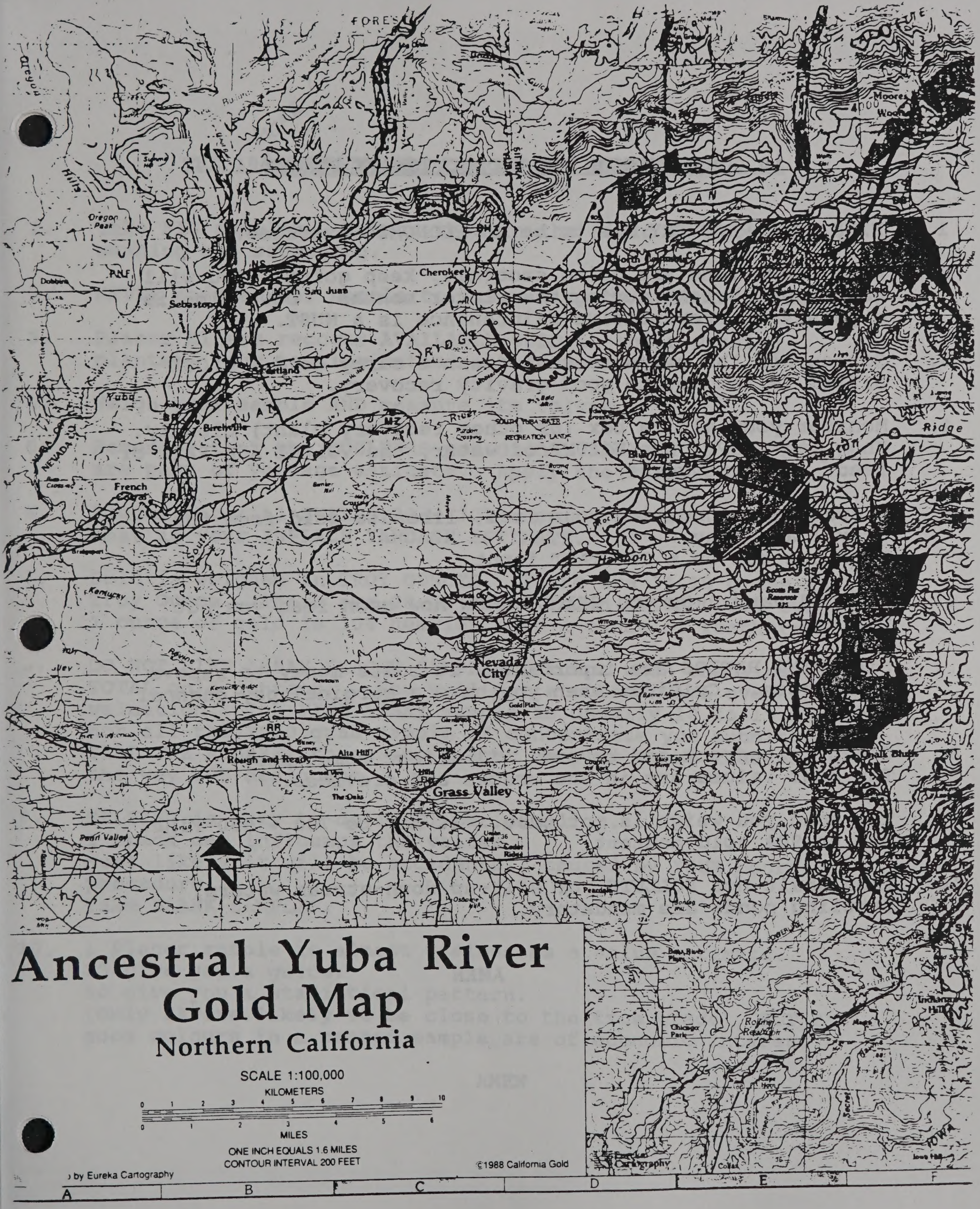
Expenditures	Cost Model 1971	Cost Model 1985	DCF Program ++ 1971	DCF Program ++ 1985	Claimant ** 1971	Claimant ** 1985
Exploration	1700	$5000	1700	$5000	1700	$5000 *
Development Costs						
Plant set-up	minimum		minimum		minimum	
Transportation	1500	2000	1500	2000	1000	2000 *
Contingency	1500	2000	1500	2000	1000	1000 *
sub-total	3000	4000	3000	4000	2000	3000
Capital Investment						
Sluice box	3400	10000	3400	10000	2500	7500
dozer (D4)	12600	37400	12600	37400	11400	34000
loader	10200	30300	10200	30300	9400	28000
pumps	900	2550	900	2550	700	2000
conc. table	600	1650	600	1650	600	1650
misc.	700	2100	700	2100	700	2100
contingency	2800	8400	2800	8400	1700	5000 *
sub-total	31200	92400	31200	92400	27000	80250*
Operating Costs						
labor	13600	40400	13600	40600	13400	40000*
fuel @8 gph	4300	12900	4300	12900	4000	12000*
maintenance	1100	3100	1100	3200	700	2000*
supplies	1700	5100	1700	5200	1500	4500*
contingency	2100	6150	2100	6250	1300	4000*
sub-total	22800	67150	22800	67650	20900	60500*
Working Capital (45% of operating costs)	10200	30450	10200	30450	9400	30250*
IRR%	.19	.16	.19	.16	.27	.19

\+ rounded to nearest $100
++ Discount Cash Flow set as close to break-even as possible, (0.1%)

* estimated
** calculated from cost indices
Price of gold: 1971= $43.50 December average
1985= $400.00 average retail price for jewelry gold

THIS PAGE
INTENTIONALLY
BLANK

Handouts



Ancestral Yuba River
Gold Map
Northern California
SCALE 1:100,000
KILOMETERS
0 1 2 3 4 5 6 7 8 9 10
0 1 2 3 4 5 6
MILES
ONE INCH EQUALS 1.6 MILES
CONTOUR INTERVAL 200 FEET
©1988 California Gold
by Eureka Cartography
Eureka Cartography
A
B
C
D
E
F
N
Oregon Peak
Dobbins
Sebastopol
South San Juan
Cherokee
Birchville
French Corral
Bridgeport
Kentucky
Yuba
Nevada City
Rough and Ready
Alta Hill
Grass Valley
The Oaks
Penn Valley
Harmony
Ridge
Moores
Chalk Bluffs
Chicago Park
Rollins Reservoir
Colfax
Iowa Hill
Indiana Hill
RECREATION LANDS
SOUTH YUBA RIVER

10 COMMANDMENTS FOR PRODUCTION IN PLACER

1. FINE GOLD RECOVERY has to be an ECONOMIC exercise.

2. When operations are going smoothly, keep going. You will get enough breaks during the winter months. Down time is expensive. Preventative maintenance is a MUST.

3. Screen off the oversize that doesn't carry gold. CLOSE SIZING is the key to good gravity recovery.

4. Monitor your sluice box (or other recovery plant) closely. Know its SLOPE, the VOLUME of water, the VOLUME AND RATE of feed. (Do not guess).

5. Prewash and condition your feed, liberate gold from the gravel and clay.
   Feed the box on a CONSTANT input.

6. "LOADS" vary. Do you REALLY KNOW how much feed you put through.

7. STRIPPING RATIOS are important. It costs money to remove overburden. Critical when you have a marginal pay channel grade.

8. Clean to bedrock and beyond. (3-4 feet sometimes). The best values are in the cracks and crevices.

9. The gold you weigh is not pure. It may be 700 - 900 Fine. 70% to 90% gold content.

10. Thou shalt not POLLUTE, or silt-up down-stream waters. Settling ponds are mandatory.

AMEN

THE 10 MORE! COMMANDMENTS FOR PLACER EXAMINATION

1. Take BIG samples. Reduce them in the field if necessary by washing and screening.

2. NEVER reduce your sample to assay size by splitting down.

3. Placer Miners recover AVAILABLE GOLD in their washing plants. Your testing method should match this.

4. Weigh your sample AND measure its volume.

5. Remember that the BOULDERS you didn't include in the sample are part of yardage estimate. Apply the BOULDER FACTOR.

6. Placer material SWELLS 20 - 30% when excavated. Include that in your INPLACE YARDAGE ESTIMATE.

7. Gold is weighed in TROY OUNCES. There are 31.10 grams in 1 Troy Ounce. There are 28.35 grams in an AVIORDUPOIS ounce. 1 ounce of gold is 10% heavier than 1 ounce of feathers.

8. DO NOT FIRE ASSAY PLACER SAMPLES (except when checking for TOTAL gold content). You will get a substantial over valuation. The FIRE ASSAY sample is usually 10 to 15 grams, not all the 2000 gram sample of concentrate you submitted. TELL THE ASSAYER these are placer samples to be treated in the way YOU designate.

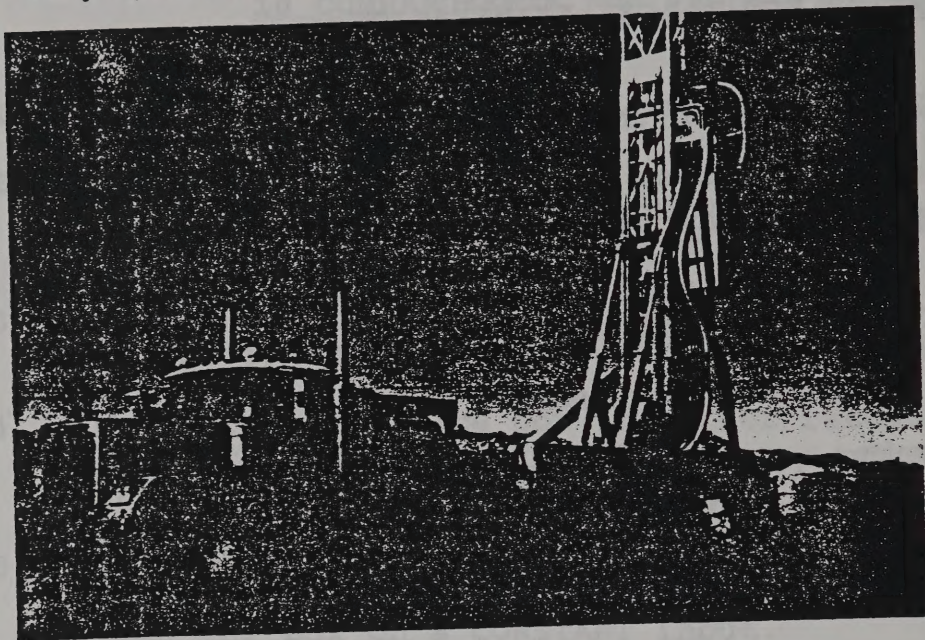
9. SALTING is easy to do in Placer sampling.
   1 speck can influence the values.
   Most likely it will be accidental, caused by YOUR poor procedure.
   Take BLANK SAMPLES.

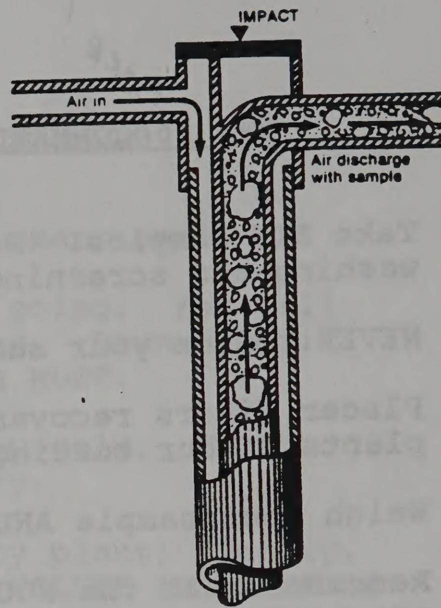
10. 1 Placer sample is almost useless as a value estimate.
    10 give you a guide.
    50 give you a statistical pattern.
    (Only 6% are likely to be close to the true grade), BUT 2 good colours in a panned sample are of economic interest.

AMEN

EQUIPMENT MAKES THE DIFFERENCE...

All of Drilling Services' Rigs are Top Head Drive, Equipped with Reverse Circulation Dual-Wall Pipe and Use a Hydraulically Operated Injection System.





Drilling Services Company maintains a wide variety of dual-wall rigs suited to most any exploration and hazardous waste drilling project in nearly any kind of formation. Each rig is kept in superb operating condition and manned by experienced personnel trained in the dual-wall method of drilling.

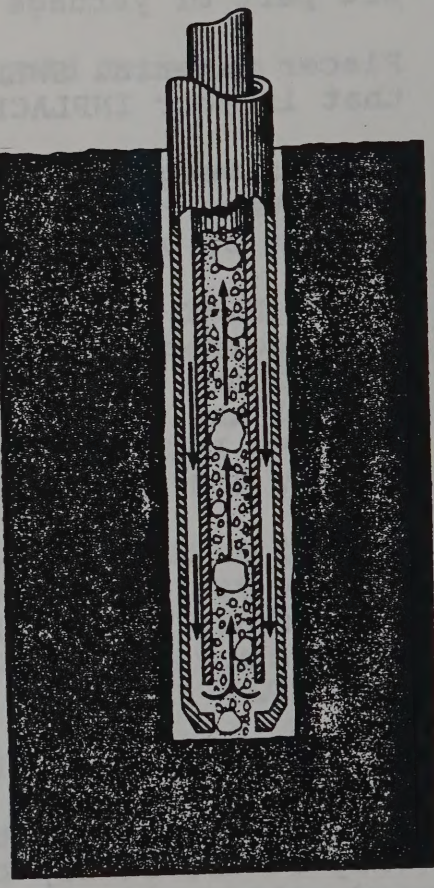
The latest addition to Drilling Services' fleet of dual-wall rigs is the AP-1000, a versatile and innovative rig that is extremely efficient in a variety of overburden soil conditions and alluvial materials.

The AP-1000 is an all purpose drill that allows the operator to convert between percussive hammer drilling, dual-wall rotary drive, drop hammer and diamond drilling with a minimum of time and labor.

Dual-Wall Rotary

The dual-wall rotary unit has a built-in water injection system which promotes easier drilling and dust suppression. It produces 4,300 ft. lbs. of torque, variable to a maximum speed of 90 RPM. The unit can be used through the inside of the hammer drill pipe for sampling and drilling in hard rock formations where the hammer pipe reaches refusal. Switching between methods takes only minutes.

As with our other dual-wall reverse circulation methods of drilling, air or water is forced down the annulus of the pipe, returning continuous and uncontaminated cuttings to the surface through the inner pipe.



DUAL-WALL REVERSE CIRCULATION OFFERS

HOW IT WORKS

The reverse circulation rotary drilling method utilizes dual-wall pipe, top drive rotation and a side inlet for injecting the drilling fluid, air or both.

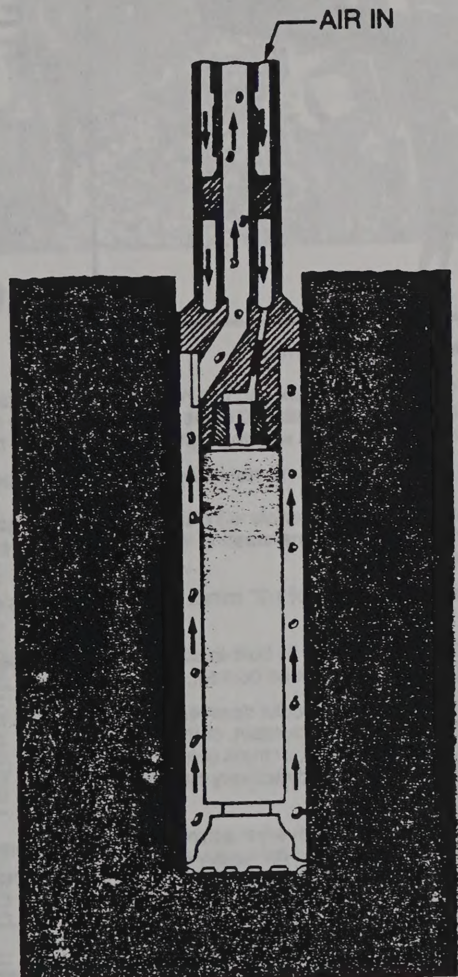
When drilling with this system, the drilling fluid is forced down the outer annulus of the dual-wall pipe to the drill bit where it is then directed to the center of the pipe. The air or drilling fluid returns the cuttings via the inner pipe at velocities in excess of 4,000 feet per minute.

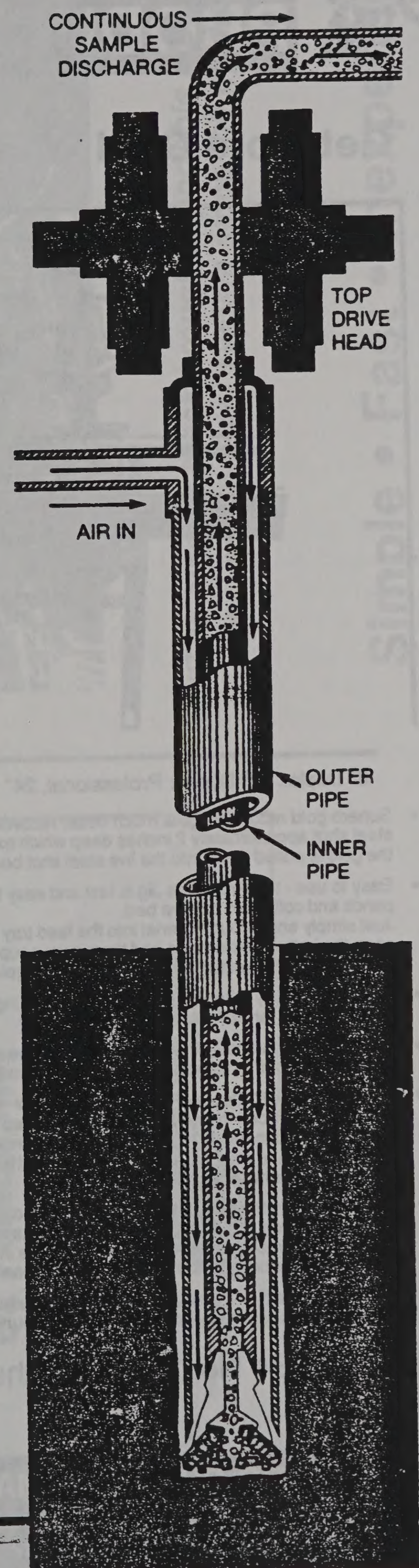
The reverse circulation rotary system uses flush-jointed drill pipe and a drill bit sub designed to fit snugly over the body of the drill bit. This unique design permits the bore hole to be cut with a minimum of clearance and so lessens the possibility of sample contamination. The configuration also minimizes the loss of air or misting in vuggy or fractured formations and will maximize the geological sample recovery in adverse conditions; conditions that are nearly impossible for rotary or diamond drills.

FOR HARD ROCK FORMATIONS

When it becomes uneconomical to drill in hard formations with either a tricone or roller button bit, the dual-wall method can still be used to advantage with a down-the-hole hammer drill.

An interchange sub is screwed onto the top of the hammer drill and then the complete unit is screwed onto the dual-wall pipe. In this configuration, the rotating/percussion action of the drill cuts the sample which travels between the wall of the hole and the outer barrel of the hammer drill. As the sample moves up the length of the hammer drill it is directed into the interchange sub openings where it is air lifted to the surface through the inner pipe.

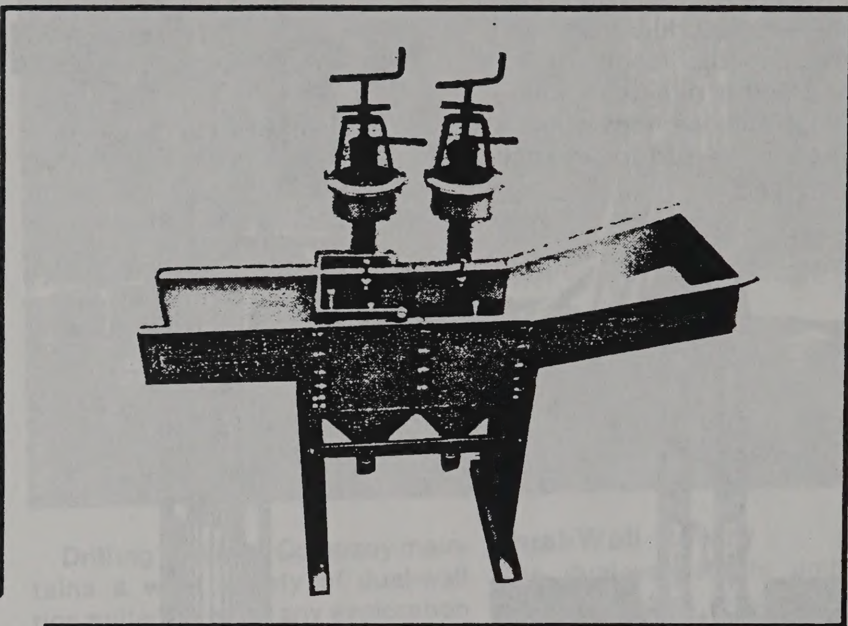




"Pro-Pulse Jigs pay for themselves with a better upgrade and concentration of your ore."

PRO-PULSE PRODUCTION GOLD RECOVERY JIG

Get More Gold With Your Own Pro-Pulse Jig



8" Duplex 12" Duplex Professional, 24" Duplex Professional

- **SUPERB GOLD RECOVERY**
- **EASY TO USE**
- **BIG FEED CAPACITY**
- **EXTRA BUILT-IN FEATURES**
- **PORTABILITY**
- **DUAL USE ABILITY**
- **LOW PRICE**

- Superb gold recovery - gets much better recovery than any sluice box and has no riffles which can plug up, instead the Pro-Pulse Jig has a live bed of 1/4" steel shot approximately 2 inches deep which expands and collapses 250 times per minute thereby pulling the fine gold into the live bed of steel shot after the gold is pulled down into the live steel shot bed it quickly works its way through the bed and to the bottom drain-off valves.
- Easy to use - the Pro-Pulse Jig is fast and easy to use and requires no electricity only 8-10 lbs. water pressure to operate the 2 pulsating heads which expands and collapses the live bed.
Just simply enter your material into the feed tray and the lighter materials will discharge out to tailings while the gold is pulled down into the live bed. Then open the bottom drain valve and there's your super fine gold! (No time-wasting riffles and carpets to pound and wash out). Also the bottom hutch valve can be partially left open to produce a continuous gold concentrate.
- Big feed capacity - the Pro-Pulse Jig has amazing thru-put capacity of 1/2" minus feed material and is extremely forgiving and easy to use. Jig has urethane wear surfaces for extra long life.
- Extra Built-in features - The Pro-Pulse Jigs have extra features such as built-in adjustable swinging gate dam which slows water and feed velocity, float gold undercurrent baffle, independent hutch adjustment, digital stroke speed counter, etc.
- Portability - the 8" Duplex Pro-Pulse Jig only weighs 55 lbs. and knocks down easily into a small parcel. It can easily be carried into the jungle, up a mountain, or into a canyon and it can easily be stowed away in a small airplane, camper, or trunk of a car. Where else could you find such a highly efficient big feed capacity gold recovery device yet with such fantastic portability?
- Dual use ability - the Pro-Pulse Jig is a highly efficient primary gold recovery machine on bulk gravel feed but it also works fantastic as a clean-up machine for heavy concentrates from sluice boxes. The Pro-Pulse Jig is fast and extremely efficient in cleaning-up concentrates which produces a clean saleable gold product.
- Low Price - the Pro-Pulse Jig is low price, is extremely efficient, can handle big feed capacity and is extremely portable. Just a few ounces of gold pays for it.

SPECIFICATIONS

	8" Duplex Portable	12" Duplex Professional	24" Duplex Professional
Feed Capacity	2 T.P.H	4 T.P.H	10 T.P.H
Water Requirements	20G.P.M @8P.S.I	30G.P.M@10P.S.I.	40G.P.M.@10P.S.I.
Stroke Adjustment	0 - 3/4"	0 - 1"	0 - 1"
Size (L. W. H)	48"x15"x36"	64"x24"x44"	78"x40"x48"
Weight	55 lbs.	210 lbs.	385 lbs.

PATENTS PENDING

Be No. 1 and join the Pros with your own Pro-Pulse Gold Recovery Jig
Order Now!

PRO-PULSE JIG MANUFACTURING LTD.
12491 Blundell Rd., Richmond, B.C., V6W 1B4 CANADA
Phone: (604) 273-4482 or (604) 685-JIGS (5447)
or (604) 669-6500 Pager 620 Fax: (604) 273-4482
Printed in Canada

Dealer

"Pro-Pulse Jigs pay for themselves with a better upgrade and concentration of your ore."

Specifications:

Rugged steel construction:

base: 2 ft. x 4 ft.
height: 48″
weight (approx.): 360 lbs.

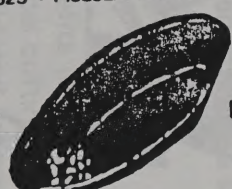
3 HP gasoline Honda* Tecumseh engine or electric motor provides all power requirements. Simple and reliable V-Belt assembly drives entire unit.

*$100 extra for Honda engine.

Write or call

E-Z PANNER, INC.

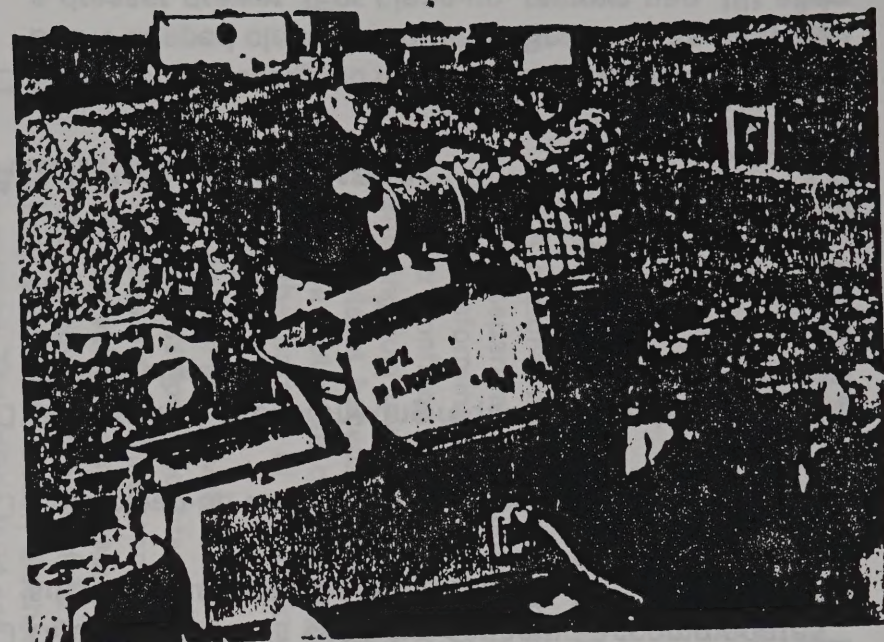
Featuring E-Z Panner Gold-Claiming Machines
Manufactured by People in the Placer-Mining Business
P. O. Box 625 • Pleasant Grove, Utah 84062



Noel S. Tanner
President
(801) 375-5080

Mark Tanner
Assistant
(801) 465-4736

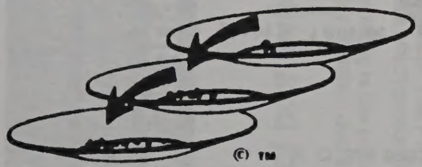
INTRODUCING the E-Z PANNER



"The biggest breakthrough in portable placer gold recovery and test system since the gold pan."

Simple • Fast • Inexpensive!

Principle of E-Z PANNER



Horizontal shaking action simulates series of miners panning from one pan to the next.

Concentrates and pans in a single operation.

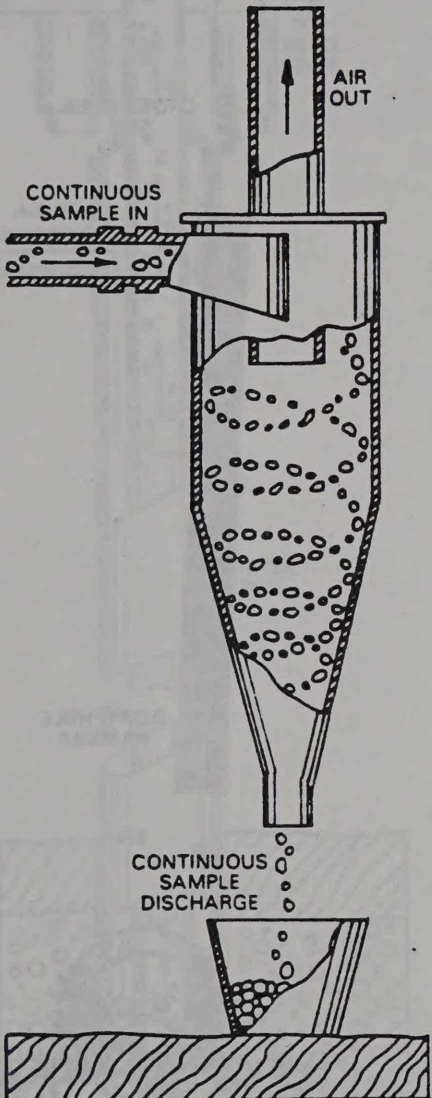
The all new E-Z PANNER features:

- ☐ Highest and most accurate recovery system on the market.
- ☐ Recovery of placer gold, including fines and ultrafines.
- ☐ Ideal for test work on drill evaluation and bulk sampling to determine results at test site with high accuracy. Eliminates bagging, tagging, transporting, and delayed inaccurate results.
- ☐ To prevent contamination from other samples, system can be cleaned quickly and easily.
- ☐ One to two yards per hour capacity. Accommodates small production placer operations.
- ☐ Cuts clean-up time on production operations 60%.
- ☐ No astroturf or cocomating required in system. Riffles remain clean.

Additional features:

- ☐ Scientifically designed integral all metal sluice, riffles, and V-shaped clean-up channel. Removes as easy as a dresser drawer. (For clean-up, remove pan, tilt and wash out through V-channel with attached water hose and spray nozzle.)
- ☐ Can be broken down for shipping. Transport in a Cessna 206.
- ☐ Can be used in dry regions. Recirculates 55 gallons of water.

CONTINUOUS REPRESENTATIVE SAMPLES



Recovery of the Geological Sample

A cyclone, which is an inertial separator without moving parts, separates particulate matter from a drilling fluid (air) by transforming the flow of an inlet stream into a double vortex confined within the cyclone. In the double vortex entering fluid spirals upward at the center of the cyclone. The particulates, because of their mass, tend to move toward the outside wall of the cyclone where they lose velocity and fall thru the bottom of the cyclone into a sample collecting container.

The Dual-Wall Reverse Circulation system of drilling delivers continuous flow, uncontaminated, representative, geological samples via the inner annulus of the dual-wall drill pipe at up hole velocities in excess of 4000 feet per minute. When the geological sample and the drilling fluid reach the surface they are simultaneously discharged via a hose directly into a cyclone. The drilling fluid and the geological sample enter the cyclone at the side/top and the cyclone's construction is such that it separates the geological sample from the drilling fluid, thus allowing the geological sample to be continuously discharged at the base, while the drilling fluid (air) is allowed to escape through the top of the cyclone. The discharged geological sample is collected in bags or buckets depending on the sample being dry or wet.

SPLITTER

Depending on geological preference the discharged geological sample can be passed through a splitter to reduce the amount of geological sample.

AUGER DRILLING

Extensively used in soils exploration, auger drilling offers many benefits in mineral exploration.

Flight Auger drilling can be an effective prospecting tool. The geological cuttings from the bit are carried to the surface on the auger flights, for grab sampling. In consolidated formations an open hole is obtained.

Hollow Stem auger drilling utilizes a flighted casing as the drill rod. The center of the bit is retractable making an open cased hole. The open casing can be utilized for standard disturbed or undisturbed sampling systems or further drilling by other drilling systems.

919 Commerce Street Fairbanks, Alaska 99701 (907) 479-0402

EFFICIENCY VERSATILITY ECONOMY

UNCONTAMINATED SAMPLES
The drill pipe is flush jointed, permitting the bore hole to be cut with minimum clearance. There is no contamination from caving formations. Also samples do not contact the walls of the bore hole on the way up.

CONTINUOUS SAMPLES
A 100% representative sample is continuously delivered to the surface as long as the drill is operating.

LARGER SAMPLE
More sample is recovered than with core drilling.

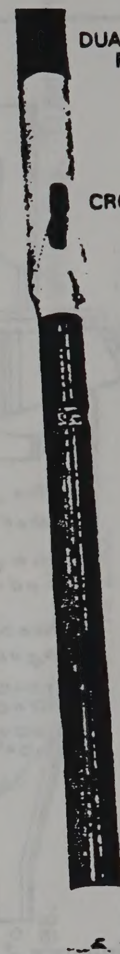
FASTER PENETRATION
Exploration proceeds up to 15 times faster than with conventional core drills.

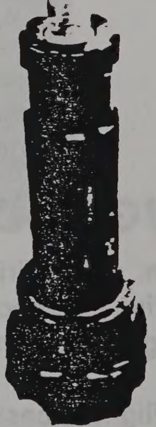
STRAIGHTER HOLES
Flush-wall pipe with less clearance results in less deviation than conventional drilling methods.

NO LOST CIRCULATION
Circulation can be maintained even while drilling in vugs, fractures, voids, and joints.

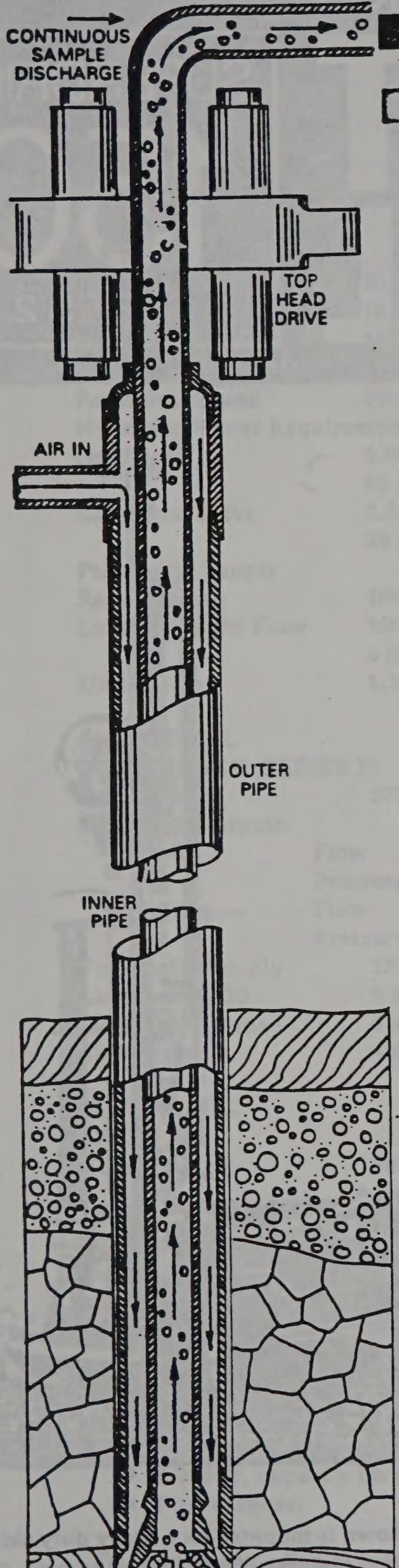
VISUAL REFERENCE
Formation changes in sample are easily noted, samples are deposited in collection container as they once lay beneath the earth.

TOTAL FOOTAGE RECOVERED
Utilizing a ground seal allows sampling to begin at the surface. No sample is lost at the start of the hole or the bottom.




DOWN HOLE HAMMER BIT


CARBIDE ROCK BIT WITH SKIRT



DUAL-WALL REVERSE-CIRCULATION

The dual-wall drill pipe consists of two tubes, one withir the other, as shown in the diagram. The hole cleaning air is forced down the annulus between the inner and outer tubes to the drill bit, where it sweeps the cuttings back up through the center tube at high velocity.

With dual-wall drilling, the drill pipe is flush jointed, creating a minimal chance for cuttings to pass up the bore hole. More importantly, no caving material can contaminate the cuttings and lost circulation zones are no problem to drill. In fact this system is often preferred in drilling through abandoned mine shafts where circulation could not be maintained by other drilling methods.

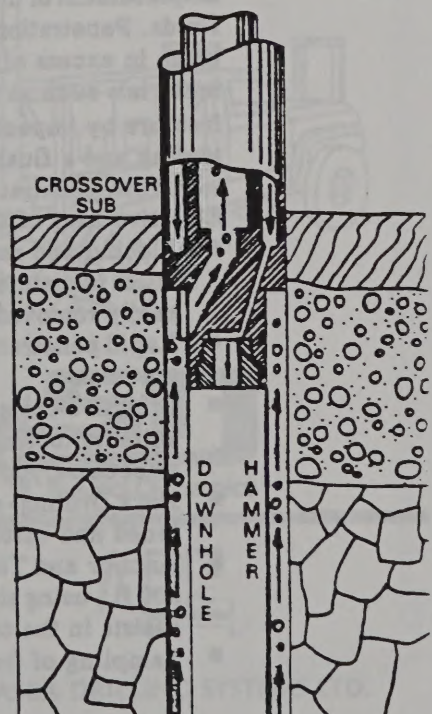
FOR HARD ROCK AND FROZEN FORMATIONS

DOWN-THE-HOLE HAMMER

When drilling hard formations becomes uneconomical with tricone bits, the down-the-hole hammer system is used.

A down-the-hole hammer is screwed onto the dual-wall pipe by using a crossover sub.

As the sample moves up along the hammer it is directed to the inner tube by the crossover sub, where the sample is moved to the surface and collected by the standard dual-wall system.



Sound Dynamics

RESONANT DRILL

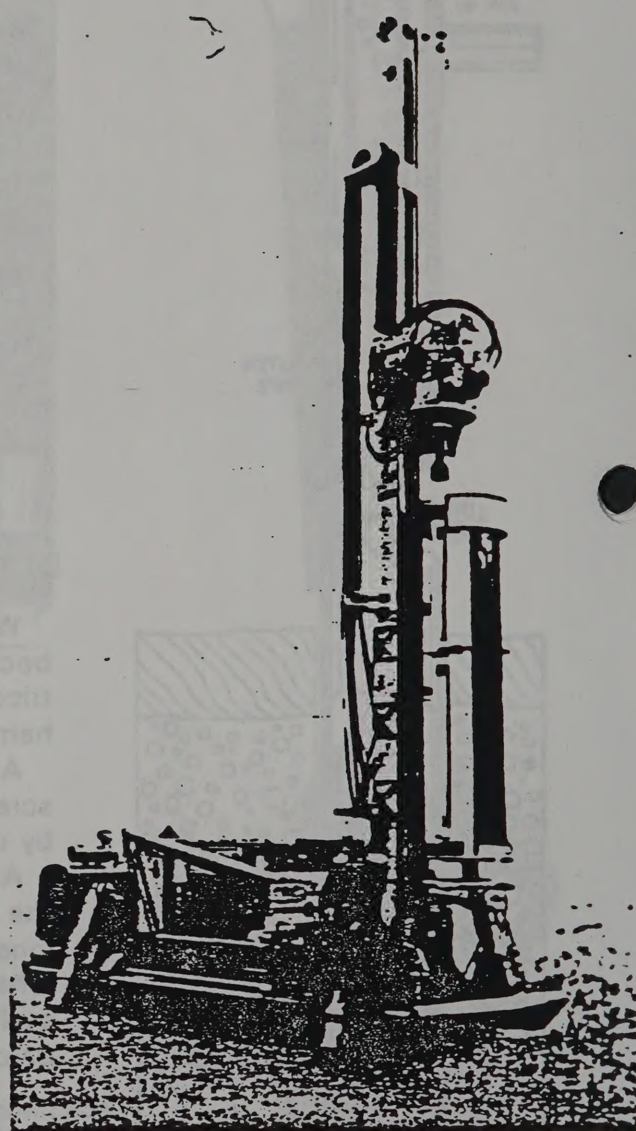
SUPERDRILL 150 SERIES 2

The Resonant Drill represents a revolutionary concept in drilling techniques which utilizes the combined actions of high frequency axial vibration with that of the more conventional rotary motion, which together, achieve effective penetration of a wide range of soil structures.

In drilling, the basic vibrational action is reinforced by direct pull-down or hoisting force, and standard rotary motion; producing a drilling capability unmatched in range and performance by any other system. Soils ranging from sands and gravels through clays and boulders to bedrock, and permafrost have been effectively cored and drilled. When drilling granular soil the action is one of "fluidisation" and displacement of the soil particles, requiring no drilling fluids. Penetration rates under these conditions are often in excess of 1 ft./sec. in more consolidated materials such as till or rock the action is one of fracture by impact. In this case the use of carbide tooling and a flushing fluid is required to achieve sustained penetration.

Effective uses of the Resonant Drill:

- **Placing and removal of overburden easing - no muds to maintain hole integrity, casing placed and easily removed.**
- **Rapid penetration - rates of 1 ft./sec. in sands and gravels.**
- **Effective coring and sampling - when fitted with suitable tooling, use of muds and fluids unnecessary.**
- **Rock Drilling - competitive drilling rates with hard faced and carbide tooling.**
- **Anchor and Tie Back Drilling - up to depths of 200 ft., using single or double wall pipe. Vibration assists in the consolidation of grout material.**
- **Sampling of frozen placer gravels.**



The Superdrill 150 (Series 2) Drill Rig consists of a BRD 150 Drill Head, Power Pack, Skid and Tower assembly, suitable for direct mounting on a truck or other carrier vehicle.

The unit shown is mounted on a heavy duty skid and has additional features of auxiliary generator, pipe handling carousel and job winch assembly.

SPECIFICATIONS ON REVERSE SIDE

SPECIFICATIONS

B.R.D. 150 DRILL HEAD

Dynamic Force Output	30,000 lbs. at 120 hz.
Operational Frequency	70 - 150 hz.
Power Input	150 hp
Torque Output	32,000 in.lbs.
Rotational Speed	60 rpm
Hydraulic Power Requirements:	
Oscillator	5,000 psi max. 65 gpm (US) max.
Rotational Drive	2,500 psi max. 28 gpm (US) max.
Pneumatic Supply Requirements	100 psi - 10 cfm
Lubricating Oil Flow	120 psi 6 (gpm)
Unit Weight	1,100 lbs.

SUPERDRILL POWER PACK (SERIES 2)

Prime Mover		170 BHP Diesel
Hydraulic Outputs:		
Main —	Flow	0 - 65 gpm (US)
	Pressure	0 - 5,000 psi
Auxiliary —	Flow	0 - 26.8 gpm (US)
	Pressure	2,500 psi
Pneumatic Supply		12 cfm at 120 psi
Lubrication Oil		6 gpm at 120 psi
Unit Dimensions		4'x8'x3'6"
Unit Weight		4,000 lbs.

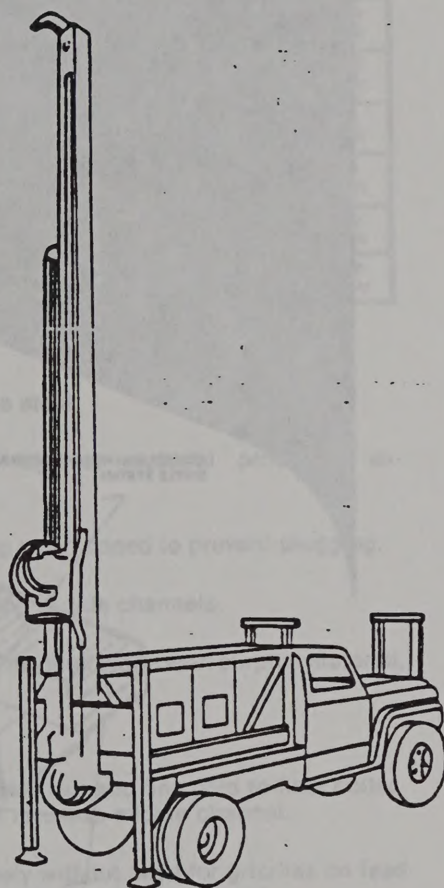
SUPERDRILL DRILL RIG

Tower	Box form with 25 ft. travel Complete with pipe joint breakout system.
Pull Down Force	11,000 lbs.
Pull Down Rate	1 ft./sec.
Skid Dimensions	4' Wide x 11'6" Long
Unit Weight	7,000 lbs.

OPTIONAL EXTRAS AVAILABLE:

- Heavy duty skid with levelling jacks
- 4,000 lb. capacity jib/winch system
- Pipe carousel

This system is available for purchase or on a lease basis as available (Canada only).



Sound Dynamics

ALASKA DRILLING SYSTEMS LTD.

Providing Resonant Drill Equipment Parts & Service to Alaska and the Pacific Northwest.

Phillips Field Road.
P.O. Box 80090

Phone: (907) 479-4210
Fairbanks, Alaska 99708

RMS-ROSS
Sluice Box

SIZE	DUMP BOX		SLUICE BOX		OVERALL	WATER VOLUME (Minimum)	WEIGHT (Estimate)
	Length	Width	Length	Width	Length		
MODEL NUMBER	FEET (metres)		FEET (metres)		FEET (metres)	U.S. Gal/Min (litres/min.)	LBS. (kilograms)
100	8 (2.44)	16 (4.88)	14 (4.27)	6 (1.83)	26 (7.93)	2500 (9,400)	15,000 (6,750)
200	10 (3.05)	18 (5.49)	18 (5.49)	9 (2.74)	34 (10.37)	3,500 (13,200)	25,000 (11,250)
300	12 (3.66)	20 (6.10)	18 (5.49)	12 (3.66)	36 (10.98)	5,000 (19,000)	35,000 (15,750)
500	14 (4.27)	20 (6.71)	18 (5.49)	16 (4.88)	38 (11.59)	8,000 (30,200)	40,00 (18,000)
700	16 (4.88)	22 (6.71)	22 (6.71)	16 (4.88)	44 (13.42)	10,000 (38,000)	45,000 (20,500)
1000	18 (5.49)	22 (6.71)	40 (12.2)	22 (6.71)	58 (17.7)	15,000 (68,250)	56,000 (25,454)

SPECIFICATIONS

DUMP BOX — large simplified constant feed — 1/2 inch plate steel.

WATER MANIFOLD — supplies pressure water in large volume against material producing excellent washing action.

PUNCH PLATE — 3/4 inch cast chrome nickel alloy steel for long wear, coned to prevent plugging.

PRIMARY — provides removal of fines for controlled washing in side channels.

FINAL — provides for final collection in case of flooding in initial stage with surplus material.

CHANNELS — 3/8 inch plate steel.

OUTSIDE — two for fines - ramp and special screen separate flour and fine gold to final collection channel. Oversize fines and excess water return to middle channel.

MIDDLE — allows all but the largest boulders to pass freely without need for grizzlies on feed end. All rocks get washed and tumbled. Coarse nuggets are collected.

GRATING — allows for safe walking and inspection, good visibility. Hinged for simple access and clean up. May be locked for security.

RIFFLES — special design for collection of fines.

FRAME — 8 inch square tubing, heavy duty reinforced to facilitate moving with construction equipment. Operated on accelerated pitch to facilitate easy movement of large sizes and volume.

CONVERSION FACTORS

1 PPM = .0001% = 1×10^{-6}
10 PMM = .001% = 1×10^{-5}
100 PMM = .01% = 1×10^{-4}
1,000 PMM = .1% = 1×10^{-3}
10,000 PMM = 1% = 1×10^{-2}
100,000 PMM = 10% = 1×10^{-1}
1,000,000 PMM = 100% = 1

.1 %
.01
.001
.0001

1). To compute \$/mgAu and ¢/mgAu

$$\frac{\text{spot price}}{\text{31 g/troy oz.}} = \$/\text{gm} * \qquad \text{ie.} \qquad \frac{430/\text{oz}}{\text{31 g/troy oz.}} = \$13.87/\text{gm} = 1.387\ ¢/\text{mg}$$

(31.1035) Actual Value

* (Move decimal point one place to the <u>LEFT</u> to convert \$/g to ¢/mg).

2). Compute value of gold per cubic yard ($\$/yd^3$)

I. ¢/mg X weight of gold in mg = ¢/sample (sample value)

ie. 1.38 ¢/mg X 2.2 mg Au = 3.05 ¢ Au

(a) $$\frac{\text{¢/sample x pan factor (See wells. pg. 89)}}{\text{number of pans taken}} = \text{value of gold}/yd^3$$

(b) $$\frac{\text{¢/sample x } 27\ ft^3/yd^3}{\text{sample vol. (in feet}^3)} = ¢/yd^3$$

ie. $$\frac{3.05\ \text{¢/sample X } 27\ ft^3/yd^3}{2\ Ft^3} = 164.7\ ¢/yd^3 = \$1.65/yd^3$$

II. (a) Compute mg Au/yd^3 ie.

$$X\ \text{mg Au}/yd^3 = \frac{\text{mg Au in sample X } 27\ ft^3/yd^3}{\text{Vol. of sample (in } ft^3)}$$

ie. $$X = \frac{2.132\ \text{mg Au X } 27\ ft^3/yd^3}{2\ ft^3} = 28.782\ \text{mg Au}/yd^3$$

28.782 mg Au X 1.387 ¢/mg
=\$ .3992/yd^3
= 40 ¢/yd^3

3). Convert PPM (from A.A assay) to Troy oz/ton ~~(gold and siver)~~

$$\frac{\text{PPM}}{34*} = \text{oz troy/ton}$$

ie 738 PPM SILVER = $\frac{738\ \text{PPM}}{34.285}$ = 21.5 troy oz/ton

*(34.285452) this figure is derived by dividing $\frac{1}{29.166} \times 1000$ for gold and silver. 1 PPM = 1 oz/ton (using an assay ton sample = 29.166g)

4). Convert PPM (from A.A. assay) to oz.AVDP/ton (base metals)

$$\frac{PPM}{31} = \text{oz.AVDP/ton}$$

(31.1035) actual value

5). To convert PPM to %

Move decimal point four places <u>LEFT</u>.
ie. .006 PPM .00000006

6). To convert % to PPM.

Move decimal point four places <u>RIGHT</u>.
ie. .006% = 60 PPM

7). To convert PPM to g/l

Move decimal point three places <u>LEFT</u>.
ie. 8,000 PPM = 8 g/l

Conversion Factors

FROM	TO	MULTIPLY BY
abamperes	amperes	$1. \times 10^{1}$
abcoulombs	statcoulombs	2.998×10^{10}
abfarads	farads	$1. \times 10^{9}$
abfarads	microfarads	$1. \times 10^{15}$
abhenries	henries	$1. \times 10^{-9}$
abhenries	millihenries	$1. \times 10^{-6}$
abohms	ohms	$1. \times 10^{-9}$
abohms	megohms	$1. \times 10^{-15}$
abvolts	volts	$1. \times 10^{-8}$
acres	sq. chains (gunters)	$1. \times 10^{1}$
acres	rods	1.60×10^{2}
acres	sq. links	$1. \times 10^{5}$
acres	hectares or sq. hectometers	4.047×10^{-1}
acres	sq. ft.	4.356×10^{4}
acres	sq. meters	4.047×10^{3}
acres	sq. miles	1.562×10^{-3}
acres	sq. yards	4.840×10^{3}
acre-feet	cu. feet	4.356×10^{4}
acre-feet	gallons	3.259×10^{5}
amperes/sq. cm.	amps/sq. in.	6.452
amperes/sq. cm.	amps/sq. meter	$1. \times 10^{4}$
amperes/sq. in.	amps/sq. cm.	1.550×10^{-1}
amperes/sq. in.	amps/sq. meter	1.550×10^{3}
amperes/sq. meter	amps/sq. cm.	1.0×10^{-4}
amperes/sq. meter	amps/sq. in.	6.452×10^{-4}
ampere-hours	coulombs	3.600×10^{3}
ampere-hours	faradays	3.731×10^{-2}
ampere-turns	gilberts	1.257
ampere-turns/cm.	amp-turns/in.	2.540
ampere-turns/cm.	amp-turns/meter	$1. \times 10^{2}$
ampere-turns/in.	amp-turns/cm.	3.937×10^{-1}
ampere-turns/in.	amp-turns/meter	3.937×10^{1}
ampere-turns/in.	gilberts/cm.	4.950×10^{-1}
ampere-turns/meter	amp-turns/cm.	$1. \times 10^{-2}$
ampere-turns/meter	amp-turns/in.	2.54×10^{-2}
ampere-turns/meter	gilberts/cm.	1.257×10^{-2}
angstrom unit	inches	3.937×10^{-9}
angstrom unit	meters	$1. \times 10^{-10}$
angstrom unit	microns or (mu)	$1. \times 10^{-4}$
ares	acres (u.s.)	2.471×10^{-2}
ares	sq. yards	1.196×10^{2}
ares	sq. meters	$1. \times 10^{2}$
astronomical unit	kilometers	1.495×10^{8}
atmospheres	tons/sq. in.	7.348×10^{-3}
atmospheres	tons/sq. foot	1.058
atmospheres	cms. of mercury (at 0°C.)	7.6×10^{1}
atmospheres	ft. of water (at 4°C.)	3.39×10^{1}
atmospheres	in of mercury (at 0°C.)	2.992×10^{1}
atmospheres	meters of mercury (at 0°C.)	7.6×10^{-1}
atmospheres	millimeters of mercury (at 0°C.)	7.6×10^{2}
atmospheres	kgs./sq. cm.	1.0333
atmospheres	kgs/sq. meter	1.0333×10^{4}
atmospheres	pounds/sq. in.	1.47×10^{1}

barrels (oil)	gallons (oil)	4.2×10^{1}
bars	atmospheres	9.869×10^{-1}
bars	dynes/sq. cm.	$1. \times 10^{6}$
bars	kgs./sq. meter	1.020×10^{4}
bars	pounds/sq. ft.	2.089×10^{3}
bars	pounds/sq. in.	1.45×10^{1}
barye	dynes/sq. cm.	1.00
bolt (u.s., cloth)	meters	3.6576×10^{1}
btu	liter-atmospheres	1.0409×10^{1}
btu	ergs	1.0550×10^{10}
btu	foot-pounds	7.7816×10^{2}
btu	gram-calories	2.52×10^{2}
btu	horsepower-hours	3.927×10^{-4}
btu	joules	1.055×10^{3}
btu	kilogram-calories	2.52×10^{-1}
btu	kilogrammeters	1.0758×10^{2}
btu	kilowatt-hours	2.928×10^{-4}
btu/hr.	ft.-pounds/sec.	2.162×10^{-1}
btu/hr.	gram-cal./sec.	7.0×10^{-2}
btu/hr.	horsepower	3.929×10^{-4}
btu/hr.	watts	2.931×10^{-1}
btu/min.	ft.-pounds/sec.	1.296×10^{1}
btu/min.	horsepower	2.356×10^{-2}
btu/min.	kilowatts	1.757×10^{-2}
btu/min.	watts	1.757×10^{1}
btu/sq. ft./min.	watts/sq. in.	1.22×10^{-1}
bucket (br. dry)	cubic cm.	1.8184×10^{4}
bushels	cubic ft.	1.2445
bushels	cubic in.	2.1504×10^{3}
bushels	cubic meters	3.524×10^{-2}
bushels	liters	3.524×10^{1}
bushels	pecks	4.0
bushels	pints (dry)	6.4×10^{1}
bushels	quarts (dry)	3.2×10^{1}

calories, gram (mean)	btu (mean)	3.9685×10^{-3}
candle/sq. cm.	lamberts	3.146
candle/sq. in.	lamberts	4.870×10^{-1}
centares	sq. meters	1.0
centigrade (degrees)	fahrenheit (degrees)	(°C x 9/5) + 32
centigrade (degrees)	kelvin (degrees)	°C + 273.18
centigrams	grams	$1. \times 10^{-2}$
centiliters	ounce (fluid) u.s.	3.382×10^{-1}
centiliters	cubic in.	6.103×10^{-1}
centiliters	drams	2.705
centiliters	liters	1.0×10^{-2}
centimeters	feet	3.281×10^{-2}
centimeters	inches	3.937×10^{-1}
centimeters	kilometers	$1. \times 10^{-5}$
centimeters	meters	$1. \times 10^{-2}$
centimeters	miles	6.214×10^{-6}
centimeters	millimeters	$1. \times 10^{1}$
centimeters	mils	3.937×10^{2}
centimeters	yards	1.094×10^{-2}
centimeters	microns	$1. \times 10^{4}$
centimeters	angstrom units	$1. \times 10^{8}$
centimeter-dynes	cn-grams	1.020×10^{-3}
centimeter-dynes	meter-kgs	1.020×10^{-8}
centimeter-dynes	pound-ft.	7.376×10^{-8}
centimeter-grams	cm.-dynes	9.807×10^{2}

mercury		
centimeters of mercury	ft. of water	4.461×10^{-1}
centimeters of mercury	kgs./sq. meter	1.36×10^{2}
centimeters of mercury	pounds/sq. ft.	2.785×10^{1}
centimeters of mercury	pounds/sq. in.	1.934×10^{-1}
centimeters/sec	feet/min.	1.969
centimeters/sec	feet/sec.	3.281×10^{-2}
centimeters/sec.	kilometers/hr.	3.6×10^{-2}
centimeters/sec.	knots	1.943×10^{-2}
centimeters/sec.	meters/min.	6.0×10^{-1}
centimeters/sec.	miles/hr.	2.237×10^{-2}
centimeters/sec.	miles/min.	3.728×10^{-4}
centimeters/sec./sec.	ft./sec./sec.	3.281×10^{-2}
centimeters/sec./sec.	kms./hr./sec.	3.6×10^{-2}
centimeters/sec./sec.	meters/sec./sec.	1.0×10^{-2}
centimeters/sec./sec.	miles/hr./sec.	2.237×10^{-2}
centipoise	gr./cm.-sec.	1.0×10^{-2}
centipoise	pound/ft.-sec.	6.72×10^{-4}
centipoise	pound/ft.-hr.	2.4
chains (gunters)	inches	7.92×10^{2}
chains (gunters)	meters	2.012×10^{1}
chains (gunters)	yards	2.2×10^{1}
circular mils	sq. cm.	5.067×10^{-6}
circular mils	sq. mils	7.854×10^{-1}
circular mils	sq. inches	7.854×10^{-7}
circumference	radians	6.283
cords	cord ft.	8.0
cord ft.	cubic ft.	1.6×10^{1}
coulombs	statcoulombs	2.998×10^{9}
coulombs	faradays	1.036×10^{-5}
coulombs/sq. cm.	coulombs/sq. in.	6.452
coulombs/sq. cm.	coulombs/sq. meter	1.0×10^{4}
coulombs/sq. in.	coulombs/sq. cm.	1.550×10^{-1}
coulombs/sq. in.	coulombs/sq. meter	1.550×10^{3}
coulombs/sq. meter	coulombs/sq. cm.	1.0×10^{-4}
coulombs/sq. meter	coulombs/sq. in.	6.452×10^{-4}
cubic centimeters	cubic ft.	3.531×10^{-5}
cubic centimeters	cubic in.	6.102×10^{-2}
cubic centimeters	cubic meters	1.0×10^{-6}
cubic centimeters	cubic yards	1.308×10^{-6}
cubic centimeters	gallons (u.s. liquid)	2.642×10^{-4}
cubic centimeters	liters	1.0×10^{-3}
cubic centimeters	pints (u.s. liquid)	2.113×10^{-3}
cubic centimeters	quarts (u.s. liquid)	1.057×10^{-3}
cubic feet	bushels (dry)	8.036×10^{-1}
cubic feet	cu. cms.	2.8320×10^{4}
cubic feet	cu. inches	1.728×10^{3}
cubic feet	cu. meters	2.832×10^{-2}
cubic feet	cu. yards	3.704×10^{-2}
cubic feet	gallons (u.s. liquid)	7.48052
cubic feet	liters	2.832×10^{1}
cubic feet	pints (u.s. liquid)	5.984×10^{1}
cubic feet	quarts (u.s. liquid)	2.992×10^{1}
cubic feet/min.	cu. cms./sec.	4.72×10^{2}
cubic feet/min.	gallons/sec.	1.247×10^{-1}
cubic feet/min.	liters/sec.	4.720×10^{-1}
cubic feet/min.	pounds water/min.	6.243×10^{1}
cubic feet/sec.	million gals./day	$6.46317 \times 10^{-}$[illegible]
cubic feet/sec.	gallons/min.	4.48831×10[illegible]
cubic inches	cu cms.	1.639×10^{1}

FROM	TO	MULTIPLY BY
cubic inches	cu. ft.	5.787×10^{-4}
cubic inches	cu. meters	1.639×10^{-5}
cubic inches	cu. yards	2.143×10^{-5}
cubic inches	gallons	4.329×10^{-3}
cubic inches	liters	1.639×10^{-2}
cubic inches	pints (u.s. liquid)	3.463×10^{-2}
cubic inches	quarts (u.s. liquid)	1.732×10^{-2}
cubic meters	bushels (dry)	2.838×10^{1}
cubic meters	cu. cms.	1.0×10^{6}
cubic meters	cu. ft.	3.531×10^{1}
cubic meters	cu. inches	6.1023×10^{4}
cubic meters	cu. yards	1.308
cubic meters	gallons (u.s. liquid)	2.642×10^{2}
cubic meters	liters	1.0×10^{3}
cubic meters	pints (u.s. liquid)	2.113×10^{3}
cubic meters	quarts (u.s. liquid)	1.057×10^{3}
cubic yards	cu. cms.	7.646×10^{5}
cubic yards	cu. ft.	2.7×10^{1}
cubic yards	cu. inches	4.6656×10^{4}
cubic yards	cu. meters	7.646×10^{-1}
cubic yards	gallons (u.s. liquid)	2.02×10^{2}
cubic yards	liters	7.646×10^{2}
cubic yards	pints (u.s. liquid)	1.6159×10^{3}
cubic yards	quarts (u.s. liquid)	8.079×10^{2}
cubic yards/min.	cubic ft./sec.	4.5×10^{-1}
cubic yards/min.	gallons/sec.	3.367
cubic yards/min.	liters/sec.	1.274×10^{1}

D

FROM	TO	MULTIPLY BY
daltons	grams	1.650×10^{-24}
days	seconds	8.64×10^{4}
days	minutes	1.44×10^{3}
days	hours	2.4×10^{1}
decigrams	grams	1.0×10^{-1}
deciliters	liters	1.0×10^{-1}
decimeters	meters	1.0×10^{-1}
degrees (angle)	quadrants	1.111×10^{-2}
degrees (angle)	radians	1.745×10^{-2}
degrees (angle)	seconds	3.6×10^{3}
degrees/sec.	radians/sec.	1.745×10^{-2}
degrees/sec.	revolutions/min.	1.667×10^{-1}
degrees/sec.	revolutions/sec.	2.778×10^{-3}
dekagrams	grams	1.0×10^{1}
dekaliters	liters	1.0×10^{1}
dekameters	meters	$10. \times 10^{1}$
drams (apoth. or troy)	ounces (avdp.)	1.3714×10^{-1}
drams (apoth. or troy)	ounces (troy)	1.25×10^{-1}
drams (u.s. fluid or apoth.)	cubic cm.	3.6967
drams	grams	1.7718
drams	grains	2.7344×10^{1}
drams	ounces	6.25×10^{-2}
dynes/sq. cm.	ergs/sq. millimeter	1.0×10^{-2}
dynes/sq. cm.	atmosphere	9.869×10^{-7}
dynes/sq. cm.	in. of mercury (at 0°C.)	2.953×10^{-5}
dynes/sq. cm.	in. of water (at 4°C.)	4.015×10^{-4}
dynes	grams	1.020×10^{-3}
dynes	joules/cm.	1.0×10^{-7}
dynes	joules/meter	1.0×10^{-5}

FROM	TO	MULTIPLY BY
ell	cm.	1.1430×10^{2}
ell	inches	4.5×10^{1}
em, pica	inch	1.67×10^{-1}
em, pica	cm.	4.233×10^{-1}
erg/sec.	dyne-cm./sec.	1.0
ergs	btu	9.486×10^{-11}
ergs	dyne-centimeters	1.0
ergs	foot-pounds	7.376×10^{-8}
ergs	gram-calories	2.389×10^{-8}
ergs	gram-cms.	1.020×10^{-3}
ergs	horsepower-hrs.	3.7250×10^{-14}
ergs	joules	1.0×10^{-7}
ergs	kg.-calories	2.389×10^{-11}
ergs	kg.-meters	1.020×10^{-8}
ergs	kilowatt-hrs.	2.773×10^{-14}
ergs	watt-hrs.	2.773×10^{-11}
ergs/sec.	btu/min.	5.668×10^{-9}
ergs/sec.	ft.-lbs./min.	4.426×10^{-6}
ergs/sec.	ft.-lbs./sec.	7.3756×10^{-8}
ergs/sec.	horsepower	1.341×10^{-10}
ergs/sec.	kg.-calories/min.	1.433×10^{-9}
ergs/sec.	kilowatts	1.0×10^{-10}

F

FROM	TO	MULTIPLY BY
farads	microfarads	$1. \times 10^{6}$
faraday/sec.	ampere (absolute)	9.65×10^{4}
faradays	ampere-hours	2.68×10^{1}
faradays	coulombs	9.649×10^{4}
fathoms	meters	1.8288
fathoms	feet	6.0
feet	centimeters	3.048×10^{1}
feet	kilometers	3.048×10^{-4}
feet	meters	3.048×10^{-1}
feet	miles (naut.)	1.645×10^{-4}
feet	miles (stat.)	1.894×10^{-4}
feet	millimeters	3.048×10^{2}
feet	mils	1.2×10^{4}
feet of water	atmospheres	2.95×10^{-2}
feet of water	in. of mercury	8.826×10^{-1}
feet of water	kgs./sq. cm.	3.048×10^{-2}
feet of water	kgs./sq. meter	3.048×10^{2}
feet of water	pounds/sq. ft.	6.243×10^{1}
feet of water	pounds/sq. in.	4.335×10^{-1}
feet/min.	cms./sec.	5.080×10^{-1}
feet/min.	feet/sec.	1.667×10^{-2}
feet/min.	kms./hr.	1.829×10^{-2}
feet/min.	meters/min.	3.048×10^{-1}
feet/min.	miles/hr.	1.136×10^{-2}
feet/sec.	cms./sec.	3.048×10^{1}
feet/sec.	kms./hr.	1.097
feet/sec.	knots	5.921×10^{-1}
feet/sec.	meters/min.	1.829×10^{1}
feet/sec.	miles/hr.	6.818×10^{-1}
feet/sec.	miles/min.	1.136×10^{-2}
feet/sec./sec.	cms./sec./sec.	3.048×10^{1}
feet/sec./sec.	kms./hr./sec.	1.097
feet/sec./sec.	meters/sec./sec.	3.048×10^{-1}
feet/sec./sec.	miles/hr./sec.	6.818×10^{-1}
feet/100 feet	per cent grade	1.0

FROM	TO	MULTIPLY BY
inches	miles	1.578×10^{-5}
inches	millimeters	2.54×10^{1}
inches	mils	1.0×10^{3}
inches	yards	2.778×10^{-2}
inches	angstrom units	2.54×10^{8}
inches	rods	5.0505×10^{-3}
inches of mercury	atmospheres	3.342×10^{-2}
inches of mercury	feet of water	1.133
inches of mercury	kgs./sq. cm.	3.453×10^{-2}
inches of mercury	kgs./sq. meter	3.453×10^{2}
inches of mercury	pounds/sq. ft.	7.073×10^{1}
inches of mercury	pounds/sq. in.	4.912×10^{-1}
in. of water (at 4 °C.)	atmospheres	2.458×10^{-3}
in. of water (at 4 °C.)	inches of mercury	7.355×10^{-2}
in. of water (at 4 °C.)	kgs./sq. cm.	2.54×10^{-3}
in. of water (at 4 °C.)	ounces/sq. in.	5.781×10^{-1}
in. of water (at 4°C.)	pounds/sq. ft.	5.204
in. of water (at 4 °C.)	pounds/sq. in.	3.613×10^{-2}
internat'l ampere	absolute amp. (u.s.)	9.998×10^{-1}
internat'l volt	absolute volt (u.s.)	1.00033
internat'l coulomb	absolute coulomb	9.99835×10^{-1}
joules	btu	9.486×10^{-4}
joules	ergs	1.0×10^{7}
joules	foot-pounds	7.736×10^{-1}
joules	kg.-calories	2.389×10^{-4}
joules	kg.-meters	1.020×10^{-1}
joules	watt-hrs.	2.778×10^{-4}
joules/cm.	grams	1.020×10^{4}
joules/cm.	dynes	1.0×10^{7}
joules/cm.	joules/meter (newtons)	1.0×10^{2}
joules/cm.	poundals	7.233×10^{2}
joules/cm.	pounds	2.248×10^{1}

K

FROM	TO	MULTIPLY BY
kilograms	dynes	9.80665×10^{5}
kilograms	grams	1.0×10^{3}
kilograms	joules/cm.	9.807×10^{-2}
kilograms	joules/meter (newtons)	9.807
kilograms	poundals	7.093×10^{1}
kilograms	pounds	2.2046
kilograms	tons (long)	9.842×10^{-4}
kilograms	tons (short)	1.102×10^{-3}
kilograms	ounces (avdp.)	3.5274×10^{1}
kilograms/cu. meter	grams/cu. cm.	1.0×10^{-3}
kilograms/cu. meter	pounds/cu. ft.	6.243×10^{-2}
kilograms/cu. meter	pounds/cu. in.	3.613×10^{-5}
kilograms/cu. meter	pounds/mil-foot	3.405×10^{-10}
kilograms/meter	pounds/ft.	6.72×10^{-1}
kilograms/sq. cm.	dynes/sq. cm.	9.80665×10^{5}
kilograms/sq. cm.	atmospheres	9.678×10^{-1}
kilograms/sq. cm.	feet of water	3.281×10^{1}
kilograms/sq. cm.	inches of mercury	2.896×10^{1}
kilograms/sq. cm.	pounds/sq. ft.	2.048×10^{3}

FROM	TO	MULTIPLY BY
kilograms/sq. meter	pounds/sq. ft.	2.048×10^{-1}
kilograms/sq. meter	pounds/sq. in.	1.422×10^{-3}
kilograms/sq. meter	dynes/sq. cm.	9.80665×10^{1}
kilograms/sq. mm.	kgs./sq. meter	1.0×10^{6}
kilogram-calories	btu	3.968
kilogram-calories	foot-pounds	3.086×10^{3}
kilogram-calories	horsepower-hrs.	1.558×10^{-3}
kilogram-calories	joules	4.183×10^{3}
kilogram-calories	kg.-meters	4.269×10^{2}
kilogram-calories	kilojoules	4.186
kilogram-calories	kilowatt-hrs.	1.163×10^{-3}
kilogram-calories/min.	ft.-lbs./sec.	5.143×10^{1}
kilogram-calories/min.	horsepower	9.351×10^{-2}
kilogram-calories/min.	kilowatts	6.972×10^{-2}
kilogram-meters	btu	9.296×10^{-3}
kilogram-meters	ergs	9.807×10^{7}
kilogram-meters	foot-pounds	7.233
kilogram-meters	joules	9.807
kilogram-meters	kg.-calories	2.342×10^{-3}
kilogram-meters	kilowatt-hrs.	2.723×10^{-6}
kilolines	maxwells	1.0×10^{3}
kiloliters	liters	1.0×10^{3}
kiloliters	cubic yards	1.308
kiloliters	cubic feet	3.5316×10^{1}
kiloliters	gallons (u.s. liquid)	2.6418×10^{2}
kilometers	centimeters	1.0×10^{5}
kilometers	feet	3.281×10^{3}
kilometers	inches	3.937×10^{4}
kilometers	meters	1.0×10^{3}
kilometers	miles (statute)	6.214×10^{-1}
kilometers	miles (nautical)	5.396×10^{-1}
kilometers	millimeters	1.0×10^{6}
kilometers	yards	1.0936×10^{3}
kilometers/hr.	cms./sec.	2.778×10^{1}
kilometers/hr.	feet/min.	5.468×10^{1}
kilometers/hr.	feet/sec.	9.113×10^{-1}
kilometers/hr.	knots	5.396×10^{-1}
kilometers/hr.	meters/min.	1.667×10^{1}
kilometers/hr.	miles/hr.	6.214×10^{-1}
kilometers/hr./sec.	cms./sec./sec.	2.778×10^{1}
kilometers/hr./sec.	ft./sec./sec.	9.113×10^{-1}
kilometers/hr./sec.	meters/sec./sec.	2.778×10^{-1}
kilometers/hr./sec.	miles/hr./sec.	6.214×10^{-1}
kilowatts	btu/min.	5.692×10^{1}
kilowatts	foot-lbs./min.	4.426×10^{4}
kilowatts	foot-lbs./sec.	7.376×10^{2}
kilowatts	horsepower	1.341
kilowatts	kg.-calories/min.	1.434×10^{1}
kilowatts	watts	1.0×10^{3}
kilowatt-hrs.	btu	3.413×10^{3}
kilowatt-hrs.	ergs	3.6×10^{13}
kilowatt-hrs.	foot-lbs.	2.655×10^{6}
kilowatt-hrs.	gram calories	8.5985×10^{5}
kilowatt-hrs.	horsepower-hours	1.341
kilowatt-hrs.	joules	3.6×10^{6}
kilowatt-hrs.	kg.-calories	8.605×10^{2}
kilowatt-hrs.	kg.-meters	3.671×10^{5}
kilowatt-hrs.	pounds of water evaporated from and at 212 °F.	3.53

foot-pounds	[illegible]	[illegible]
foot-pounds	kilowatt-hrs.	3.766×10^{-7}
foot-pounds/min.	btu/min.	1.286×10^{-3}
foot-pounds/min.	foot-pounds/sec.	1.667×10^{-2}
foot-pounds/min.	horsepower	3.030×10^{-5}
foot-pounds/min.	kg.-calories/min.	3.241×10^{-4}
foot-pounds/min.	kilowatts	2.260×10^{-5}
foot-pounds/sec.	btu/hr.	4.6263
foot-pounds/sec.	btu/min.	7.717×10^{-2}
foot-pounds/sec.	horsepower	1.818×10^{-3}
foot-pounds/sec.	kg.-calories/min.	1.945×10^{-2}
foot-pounds/sec.	kilowatts	1.356×10^{-3}
furlongs	miles (u.s.)	1.25×10^{-1}
furlongs	rods	4.0×10^{1}
furlongs	feet	6.6×10^{2}
furlongs	meters	2.0117×10^{2}

G

gallons	cu. cms.	3.785×10^{3}
gallons	cu. feet	1.377×10^{-1}
gallons	cu. inches	2.31×10^{2}
gallons	cu. meters	3.785×10^{-3}
gallons	cu. yards	4.951×10^{-3}
gallons	liters	3.785
gallons (liq. br. imp.)	gallons (u.s. liquid)	1.20095
gallons (u.s.)	gallons (imp.)	8.3267×10^{-1}
gallons of water	pounds of water	8.337
gallons/min.	cu. feet/sec.	2.228×10^{-3}
gallons/min.	liters/sec.	6.308×10^{-2}
gallons/min.	cu. feet/hr.	8.0208
gausses	lines/sq. in.	6.452
gausses	webers/sq. cm.	1.0×10^{-8}
gausses	webers/sq. in.	6.452×10^{-8}
gausses	webers/sq. meter	1.0×10^{-4}
gausses	amp.-turn/cm.	7.958×10^{-1}
gausses	gilbert/cm.	1.0
gilberts	ampere-turns	7.958×10^{-1}
gilberts/cm.	ampere-turns/cm.	7.958×10^{-1}
gilberts/cm.	ampere-turns/in.	2.021
gilberts/cm.	ampere-turns/meter	7.958×10^{1}
gills (british)	cubic cm.	1.4207×10^{2}
gills (u.s.)	cubic cm.	1.18295×10^{2}
gills (u.s.)	liters	1.183×10^{-1}
gills (u.s.)	pints (liq.)	2.5×10^{-1}
grade	radian	1.571×10^{-2}
grains	drams (avdp.)	3.657×10^{-2}
grains (troy)	grains (avdp.)	1.0
grains (troy)	grams	6.48×10^{-2}
grains (troy)	ounces (avdp.)	2.0833×10^{-3}
grains (troy)	pennyweight (troy)	4.167×10^{-2}
grains/u.s. gallon	parts/million	1.7118×10^{1}
grains/u.s. gallon	pounds/million gallons	1.4286×10^{2}
grains/imp. gallon	parts/million	1.4286×10^{1}
grams	dynes	9.807×10^{2}
grams	grains (troy)	1.543×10^{1}
grams	joules/cm.	9.807×10^{-5}
grams	joules/meter (newtons)	9.807×10^{-3}
grams	kilograms	1.0×10^{-3}
grams	milligrams	1.0×10^{3}
grams	ounces (avdp.)	3.527×10^{-2}
grams	ounces (troy)	3.215×10^{-2}
[illegible]	[illegible]	[illegible]
grams/cu. cm.	pounds/cu. in.	3.613×10^{-2}
grams/cu. cm.	pounds/mil-foot	3.405×10^{-7}
grams/liter	grains/gal.	5.8417×10^{1}
grams/liter	pounds/1,000 gal.	8.345
grams/liter	pounds/cu. ft.	$6.2427 \times 10^{[illegible]}$
grams/sq. cm.	pounds/sq. ft.	2.0481
gram-calories	btu	3.9683×10^{-3}
gram-calories	ergs	4.184×10^{7}
gram-calories	foot-pounds	3.086
gram-calories	horsepower-hrs.	1.5596×10^{-3}
gram-calories	kilowatt-hrs.	1.162×10^{-6}
gram-calories	watt-hrs.	1.162×10^{-3}
gram-calories/sec.	btu/hr.	1.4286×10^{1}
gram-centimeters	btu	9.297×10^{-8}
gram-centimeters	ergs	9.807×10^{2}
gram-centimeters	joules	9.807×10^{-5}
gram-centimeters	kg.-calories	2.343×10^{-8}
gram-centimeters	kg.-meters	1.0×10^{-5}

hand	cm.	1.016×10^{1}
hectares	acres	2.471
hectares	sq. feet	1.076×10^{5}
hectograms	grams	1.0×10^{2}
hectoliters	liters	1.0×10^{2}
hectometers	meters	1.0×10^{2}
hectowatts	watts	1.0×10^{2}
henries	millihenries	1.0×10^{3}
hogsheads (british)	cubic ft.	1.0114×10^{1}
hogsheads (u.s.)	cubic ft.	8.42184
hogsheads (u.s.)	gallons (u.s.)	6.3×10^{1}
horsepower	btu/min	$4.244 \times 10^{[illegible]}$
horsepower	foot-lbs./min.	3.3×10^{4}
horsepower	foot-lbs./sec.	5.50×10^{2}
horsepower (metric)	horsepower	9.863×10^{-1}
horsepower	horsepower (metric)	1.014
horsepower	kg.-calories/min.	1.068×10^{1}
horsepower	kilowatts	7.457×10^{-1}
horsepower	watts	7.457×10^{2}
horsepower (boiler)	btu/hr.	3.352×10^{4}
horsepower (boiler)	kilowatts	9.803
horsepower-hours	btu	2.547×10^{3}
horsepower-hours	ergs	2.6845×10^{13}
horsepower-hours	foot-lbs.	1.98×10^{6}
horsepower-hours	gram-calories	6.4119×10^{5}
horsepower-hours	joules	2.684×10^{6}
horsepower-hours	kg-calories	6.417×10^{2}
horsepower-hours	kg.-meters	2.737×10^{5}
horsepower-hours	kilowatt-hrs	7.457×10^{-1}
hours	days	4.167×10^{-2}
hours	weeks	5.952×10^{-3}
hours	seconds	3.6×10^{3}
hundredwgts(long)	pounds	1.12×10^{2}
hundredwgts(long)	tons (long)	5.0×10^{-2}
hundredwgts(long)	kilograms	5.08023×10^{1}
hundredwgts(short)	tons (metric)	4.53592×10^{-2}
hundredwgts(short)	tons (long)	4.46429×10^{-2}
hundredwgts(short)	kilograms	4.53592×10^{1}

inches	centimeters	2.540
inches	meters	2.540×10^{-2}

knots	feet/sec.	1.689
knots	cm./sec.	5.148×10^{1}

lambert	candle/sq. cm.	3.183×10^{-1}
lambert	candle/sq. in.	2.054
league	miles (approx.)	3.0
light year	miles	5.9×10^{12}
light year	kilometers	9.46091×10^{12}
lines/sq. cm.	gausses	1.0
lines/sq. in.	gausses	1.55×10^{-1}
lines/sq. in.	webers/sq. cm.	1.55×10^{-9}
lines/sq. in.	webers/sq. in.	1.0×10^{-8}
lines/sq. in.	webers/sq. meter	1.55×10^{-5}
links (engineers)	inches	1.2×10^{1}
links (surveyors)	inches	7.92
liters	bushels (u.s. dry)	2.838×10^{-2}
liters	cu. cm.	1.0×10^{3}
liters	cu. ft.	3.531×10^{-2}
liters	cu. inches	6.102×10^{1}
liters	cu. meters	1.0×10^{-3}
liters	cu. yards	1.308×10^{-3}
liters	gallons (u.s. liquid)	2.642×10^{-1}
liters	pints (u.s. liquid)	2.113
liters	quarts (u.s. liquid)	1.057
liters/min.	cu. ft./sec.	5.886×10^{-4}
liters/min.	gals./sec.	4.403×10^{-3}
$\log_{10} n$	$\ln n$	2.303
$\ln n$	$\log_{10} n$	4.343×10^{-1}
lumen	spherical candle power	7.958×10^{-2}
lumen/sq. ft.	foot-candles	1.0
lumen/sq. ft.	lumen-sq. meter	1.076×10^{1}
lux	foot-candles	9.29×10^{-2}

M

maxwells	kilolines	1.0×10^{-3}
maxwells	webers	1.0×10^{-8}
megalines	maxwells	1.0×10^{6}
megohms	microhms	1.0×10^{12}
megohms	ohms	1.0×10^{6}
megmhos/cubic cm.	abmhos/cubic cm.	1.0×10^{-3}
megmhos/cubic cm.	megmhos/cubic in.	2.54
megmhos/cubic cm.	mhos/mil. ft.	1.662×10^{-1}
megmhos/in. cube	megmhos/cubic cm.	3.937×10^{-1}
meters	angstrom units	1.0×10^{10}
meters	centimeters	1.0×10^{2}
meters	fathoms	5.4681×10^{-1}
meters	feet	3.281
meters	inches	3.937×10^{1}
meters	kilometers	1.0×10^{-3}
meters	miles (nautical)	5.396×10^{-4}
meters	miles (statute)	6.214×10^{-4}
meters	millimeters	1.0×10^{3}
meters	yards	1.094
meters/min.	cms./sec.	1.667
meters/min.	feet/min.	3.281
meters/min.	feet/sec.	5.468×10^{-2}
meters/sec.	feet/min.	1.966×10^{1}
meters/sec.	feet/sec.	3.281
meters/sec.	kilometers/hr.	3.6
meters/sec.	kilometers/min.	6.0×10^{-2}
meters/sec.	miles/hr.	2.237
meters/sec.	miles/min.	3.728×10^{-2}
meters/sec./sec.	cms./sec./sec.	1.0×10^{2}
meters/sec./sec.	ft./sec./sec.	3.281
meters/sec./sec.	kms./hr./sec	3.6
meters/sec./sec	miles/hr./sec	2.237
meter-kilograms	cm.-dynes	9.807×10^{7}
meter-kilograms	cm.-grams	1.0×10^{5}
meter-kilograms	pound-feet	7.233
microfarads	abfarads	1.0×10^{-15}
microfarads	farads	1.0×10^{-6}
microfarads	statfarads	9.0×10^{5}
micrograms	grams	1.0×10^{-6}
microhms	abohms	1.0×10^{3}
microhms	megohms	1.0×10^{-12}
microhms	ohms	1.0×10^{-6}
microliters	liters	1.0×10^{-6}
micromicrons	meters	1.0×10^{-12}
microns	meters	1.0×10^{-6}
miles (nautical)	feet	6.076×10^{3}
miles (nautical)	kilometers	1.853
miles (nautical)	meters	1.853×10^{3}
miles (nautical)	miles (statute)	1.1516
miles (nautical)	yards	2.0254×10^{3}
miles (statute)	centimeters	1.609×10^{5}
miles (statute)	feet	5.280×10^{3}
miles (statute)	inches	6.336×10^{4}
miles (statute)	kilometers	1.609
miles (statute)	meters	1.609×10^{3}
miles (statute)	miles (nautical)	8.684×10^{-1}
miles (statute)	yards	1.760×10^{3}
miles (statute)	light years	1.69×10^{-13}
miles/hr.	cms./sec.	4.470×10^{1}
miles/hr.	ft./min.	8.8×10^{1}
miles/hr.	ft./sec.	1.467
miles/hr.	kms./hr.	1.6093
miles/hr.	kms./min.	2.682×10^{-2}
miles/hr.	knots	8.684×10^{-1}
miles/hr.	meters/min.	2.682×10^{1}
miles/hr.	miles/min.	1.667×10^{-2}
miles/hr./sec.	cms./sec./sec.	4.47×10^{1}
miles/hr./sec.	ft./sec./sec	1.467
miles/hr./sec.	kms./hr./sec	1.6093
miles/hr./sec.	meters/sec./sec.	4.47×10^{-1}
miles/min.	cms./sec.	2.682×10^{3}
miles/min.	feet/sec.	8.8×10^{1}
miles/min.	kms./min.	1.6093
miles/min.	knots/min.	8.684×10^{-1}
miles/min.	miles/hr.	6.0×10^{1}
milliers	kilograms	1.0×10^{3}
millimicrons	meters	1.0×10^{-9}
milligrams	grains	1.5432×10^{-2}
milligrams	grams	1.0×10^{-3}
milligrams/liter	parts/million	1.0
millihenries	henries	1.0×10^{-3}
milliliters	liters	1.0×10^{-3}
millimeters	centimeters	1.0×10^{-1}
millimeters	feet	3.281×10^{-3}
millimeters	inches	3.937×10^{-2}
millimeters	kilometers	1.0×10^{-6}
millimeters	meters	1.0×10^{-3}
millimeters	miles	6.214×10^{-7}

FROM	TO	MULTIPLY BY
millimeters	mils	3.937×10^{1}
millimeters	yards	1.094×10^{-3}
million gals./day	cu. ft./sec.	1.54723
mils	centimeters	2.54×10^{-3}
mils	feet	8.333×10^{-5}
mils	inches	1.0×10^{-3}
mils	kilometers	2.54×10^{-8}
mils	yards	2.778×10^{-5}
miner's inches	cu. ft./min.	1.5
minims (british)	cubic cm.	5.9192×10^{-2}
minims (u.s. fluid)	cubic cm.	6.1612×10^{-2}
minutes (angles)	degrees	1.667×10^{-2}
minutes (angles)	quadrants	1.852×10^{-4}
minutes (angles)	radians	2.909×10^{-4}
minutes (angles)	seconds	6.0×10^{1}
minutes (time)	weeks	9.9206×10^{-5}
minutes (time)	days	6.944×10^{-4}
minutes (time)	hours	1.667×10^{-2}
minutes (time)	seconds	6.0×10^{1}
myriagrams	kilograms	1.0×10^{1}
myriameters	kilometers	1.0×10^{1}
myriawatts	kilowatts	1.0×10^{1}

N

FROM	TO	MULTIPLY BY
nails	inches	2.25
newtons	dynes	1.0×10^{5}

O

FROM	TO	MULTIPLY BY
ohm (international)	ohm (absolute)	1.0005
ohms	megohms	1.0×10^{-6}
ohms	microhms	1.0×10^{6}
ounces	drams	8.0
ounces	grains	4.375×10^{2}
ounces	grams	2.8349×10^{1}
ounces	pounds	6.25×10^{-2}
ounces	ounces (troy)	9.115×10^{-1}
ounces	tons (long)	2.790×10^{-5}
ounces	tons (short)	3.125×10^{-5}
ounces (fluid)	cu. inches	1.805
ounces (fluid)	liters	2.957×10^{-2}
ounces (troy)	grains	4.80×10^{2}
ounces (troy)	grams	3.1103×10^{1}
ounces (troy)	ounces (avdp.)	1.097
ounces (troy)	pennyweights (troy)	2.0×10^{1}
ounces (troy)	pounds (troy)	8.333×10^{-2}
ounce/sq. in.	dynes/sq. cm.	4.309×10^{3}
ounce/sq. in.	pounds/sq. in.	6.25×10^{-2}

P

FROM	TO	MULTIPLY BY
pace	inches	3.0×10^{1}
parsec	miles	1.9×10^{13}
parsec	kilometers	3.084×10^{13}
parts/million	grains/u.s. gal	5.84×10^{-2}
parts/million	grains/imp. gal.	7.016×10^{-2}
parts/million	pounds/million gal.	8.345
pecks (british)	cubic inches	5.546×10^{2}

FROM	TO	MULTIPLY BY
pennyweights (troy)	ounces (troy)	5.0×10^{-2}
pennyweights (troy)	grams	1.555
pennyweights (troy)	pounds (troy)	4.1667×10^{-3}
pints (dry)	cubic inches	3.36×10^{1}
pints (dry)	bushels	1.5625×10^{-2}
pints (dry)	quarts	5.0×10^{-1}
pints (dry)	liters	5.5059×10^{-1}
pints (liquid)	cubic cms.	4.732×10^{2}
pints (liquid)	cubic ft.	1.671×10^{-2}
pints (liquid)	cubic inches	2.887×10^{1}
pints (liquid)	cubic meters	4.732×10^{-4}
pints (liquid)	cubic yards	6.189×10^{-4}
pints (liquid)	gallons	1.25×10^{-1}
pints (liquid)	liters	4.732×10^{-1}
pints (liquid)	quarts (liquid)	5.0×10^{-1}
planck's quantum	erg-seconds	6.624×10^{-27}
poise	gram/cm.-sec.	1.0
pounds (avdp.)	ounces (troy)	1.4583×10^{1}
poundals	dynes	1.3826×10^{4}
poundals	grams	1.41×10^{1}
poundals	joules/cm.	1.383×10^{-3}
poundals	joules/meter (newtons)	1.383×10^{-1}
poundals	kilograms	1.41×10^{-2}
poundals	pounds	3.108×10^{-2}
pounds	drams	2.56×10^{2}
pounds	dynes	4.448×10^{5}
pounds	grains	7.0×10^{3}
pounds	grams	4.5359×10^{2}
pounds	joules/cm.	4.448×10^{-2}
pounds	joules/meter (newtons)	4.448
pounds	kilograms	4.536×10^{-1}
pounds	ounces	1.6×10^{1}
pounds	ounces (troy)	1.458×10^{1}
pounds	poundals	3.217×10^{1}
pounds	pounds (troy)	1.21528
pounds	tons (short)	5.0×10^{-4}
pounds (troy)	grains	5.760×10^{3}
pounds (troy)	grams	3.7324×10^{2}
pounds (troy)	ounces (advp.)	1.3166×10^{1}
pounds (troy)	ounces (troy)	1.2×10^{1}
pounds (troy)	pennyweights (troy)	2.4×10^{2}
pounds (troy)	pounds (advp.)	8.2286×10^{-1}
pounds (troy)	tons (long)	3.6735×10^{-4}
pounds (troy)	tons (metric)	3.7324×10^{-4}
pounds (troy)	tons (short)	4.1143×10^{-4}
pounds of water	cu. ft.	1.602×10^{-2}
pounds of water	cu. inches	2.768×10^{1}
pounds of water	gallons	1.198×10^{-1}
pounds of water/min.	cu. ft./sec.	2.670×10^{-4}
pound-feet	cm.-dynes	1.356×10^{7}
pound-feet	cm.-grams	1.3825×10^{4}
pound-feet	meter-kgs.	1.383×10^{-1}
pounds/cu. ft.	grams/cu. cm.	1.602×10^{-2}
pounds/cu. ft.	kgs./cu. meter	1.602×10^{1}
pounds/cu. ft.	pounds/cu. inches	5.787×10^{-4}
pounds/cu. ft.	pounds/mil-foot	5.456×10^{-9}
pounds/cu. in.	grams/cu. cm.	2.768×10^{1}
pounds/cu. in.	kgs./cu. meter	2.768×10^{4}
pounds/cu. in.	pounds/cu. ft.	1.728×10^{3}
pounds/cu. in.	pounds/mil-foot	9.425×10^{-6}

pounds/sq. in.	feet of water	2.307
pounds/sq in.	inches of mercury	2.036
pounds/sq.in.	kgs./sq. meter	7.031×10^{2}
pounds/sq. in.	pounds/sq ft.	1.44×10^{2}
pounds/sq. in.	short tons/sq. ft.	7.2×10^{-2}
pounds/sq. in.	kgs./sq. cm.	7.03×10^{-2}

quadrants (angle)	degrees	9.0×10^{1}
quadrants (angle)	minutes	5.4×10^{3}
quadrants (angle)	radians	1.571
quadrants (angle)	seconds	3.24×10^{5}
quarts (dry)	cu inches	6.72×10^{1}
quarts (liquid)	cu. cms.	9.464×10^{2}
quarts (liquid)	cu. ft.	3.342×10^{-2}
quarts (liquid)	cu. inches	5.775×10^{1}
quarts (liquid)	cu. meters	9.464×10^{-4}
quarts (liquid)	cu. yards	1.238×10^{-3}
quarts (liquid)	gallons	2.5×10^{-1}
quarts (liquid)	liters	9.463×10^{-1}

radians	degrees	5.7296×10^{1}
radians	minutes	3.438×10^{3}
radians	quadrants	6.366×10^{-1}
radians	seconds	2.063×10^{5}
radians/sec.	degrees/sec.	5.7296×10^{1}
radians/sec.	revolutions/min.	9.549
radians/sec.	revolution/sec.	1.592×10^{-1}
radians/sec./sec.	revs./min./min.	5.7296×10^{2}
radians/sec./sec.	revs./min./sec.	9.549
radians/sec /sec.	revs /sec./sec.	1.592×10^{-1}
reams	sheets	5.0×10^{2}
revolutions	degrees	3.60×10^{2}
revolutions	quadrants	4.0
revolutions	radians	6.283
revolutions/min.	degrees/sec.	6.0
revolutions/min.	radians/sec.	1.047×10^{-1}
revolutions/min.	revs./sec.	1.667×10^{-2}
revs./min./min.	radians/sec./sec.	1.745×10^{-3}
revs./min./min.	revs./min./sec.	1.667×10^{-2}
revs./min./min.	revs./sec./sec.	2.778×10^{-4}
revolutions/sec.	degrees/sec.	3.6×10^{2}
revolutions/sec.	radians/sec.	6.283
revolutions/sec.	revs./min.	6.0×10^{1}
revs./sec /sec.	radians/sec./sec.	6.283
revs./sec./sec.	revs./min./min.	3.6×10^{3}
revs./sec./sec.	revs./min./sec.	6.0×10^{1}
rods	chains (gunters)	2.5×10^{-1}
rods	meters	5.029
rods (surveyors' meas.)	yards	5.5
rods	feet	1.65×10^{1}
rods	inches	1.98×10^{2}
rods	miles	3.125×10^{-3}
rope	feet	2.0×10^{1}

scruples	grains	2.0×10^{1}
seconds (angle)	degrees	2.778×10^{-4}
seconds (angle)	minutes	1.667×10^{-2}
slugs	pounds	3.217×10^{1}
sphere (solid angle)	steradians	1.257×10^{1}
square centimeters	circular mils	1.973×10^{5}
square centimeters	sq. feet	1.076×10^{-3}
square centimeters	sq. inches	1.550×10^{-1}
square centimeters	sq. meters	1.0×10^{-4}
square centimeters	sq. miles	3.861×10^{-11}
square centimeters	sq. millimeters	1.0×10^{2}
square centimeters	sq. yards	1.196×10^{-4}
square degrees	steradians	3.0462×10^{-4}
square feet	acres	2.296×10^{-5}
square feet	circular mils	1.833×10^{8}
square feet	sq. cms.	9.29×10^{2}
square feet	sq. inches	1.44×10^{2}
square feet	sq. meters	9.29×10^{-2}
square feet	sq. miles	3.587×10^{-8}
square feet	sq. millimeters	9.29×10^{4}
square feet	sq. yards	1.111×10^{-1}
square inches	circular mils	1.273×10^{6}
square inches	sq. cms.	6.452
square inches	sq. ft.	6.944×10^{-3}
square inches	sq. millimeters	6.452×10^{2}
square inches	sq. mils	1.0×10^{6}
square inches	sq. yards	7.716×10^{-4}
square kilometers	acres	2.471×10^{2}
square kilometers	sq. cms.	1.0×10^{10}
square kilometers	sq. ft.	1.076×10^{7}
square kilometers	sq. inches	1.550×10^{9}
square kilometers	sq. meters	1.0×10^{6}
square kilometers	sq. miles	3.861×10^{-1}
square kilometers	sq. yards	1.196×10^{6}
square meters	acres	2.471×10^{-4}
square meters	sq. cms.	1.0×10^{4}
square meters	sq. ft.	1.076×10^{1}
square meters	sq. inches	1.55×10^{3}
square meters	sq. miles	3.861×10^{-7}
square meters	sq. millimeters	1.0×10^{6}
square meters	sq. yards	1.196
square miles	acres	6.40×10^{2}
square miles	sq. ft.	2.788×10^{7}
square miles	sq. kms.	2.590
square miles	sq. meters	2.590×10^{6}
square miles	sq. yards	3.098×10^{6}
square millimeters	circular mils	1.973×10^{3}
square millimeters	sq. cms.	1.0×10^{-2}
square millimeters	sq. ft.	1.076×10^{-5}
square millimeters	sq. inches	1.55×10^{-3}
square mils	circular mils	1.273
square mils	sq. cms.	6.452×10^{-6}
square mils	sq. inches	1.0×10^{-6}
square yards	acres	2.066×10^{-4}
square yards	sq. cms.	8.361×10^{3}
square yards	sq. ft.	9.0
square yards	sq. inches	1.296×10^{3}
square yards	sq. meters	8.361×10^{-1}
square yards	sq. miles	3.228×10^{-7}
square yards	sq. millimeters	8.361×10^{5}
steradians	spheres	7.958×10^{-2}
steradians	hemispheres	1.592×10^{-1}
steradians	spherical right angles	6.366×10^{-1}
steradians	square degrees	3.283×10^{3}
steres	liters	9.99973×10^{2}

temperature (°C.) +273	absolute temperature (°K.)	1.0
temperature (°C.) +17.78	temperature (°F.)	1.8
temperature (°F.) +460	absolute temperature (°R.)	1.0
temperature (°F.) − 32	temperature (°C.)	5/9
tons (long)	kilograms	1.016×10^{3}
tons (long)	pounds	2.24×10^{3}
tons (long)	tons (short)	1.12
tons (metric)	kilograms	1.0×10^{3}
tons (metric)	pounds	2.205×10^{3}
tons (short)	kilograms	9.0718×10^{2}
tons (short)	ounces	3.2×10^{4}
tons (short)	ounces (troy)	2.9166×10^{4}
tons (short)	pounds	2.0×10^{3}
tons (short)	pounds (troy)	2.43×10^{3}
tons (short)	tons (long)	8.9287×10^{-1}
tons (short)	tons (metric)	9.078×10^{-1}
tons (short)/sq. ft.	kgs./sq. meter	9.765×10^{3}
tons (short)/sq. ft.	pounds/sq.in.	1.389×10^{1}
tons (short)/sq. in.	kgs./sq. meter	1.406×10^{6}
tons (short)/sq.in.	pounds/sq. in.	2.0×10^{3}
tons of water/24 hrs.	pounds of water/hr.	8.333×10^{1}
tons of water/24 hrs.	gallons/min.	1.6643×10^{-1}
tons of water/24 hrs.	cu. ft./hr.	1.3349

V

volt/inch	abvolts/cm.	3.937×10^{7}
volt/inch	volt/cm.	3.937×10^{-1}
volt (absolute)	statvolts	3.336×10^{-3}
volts	abvolts	1.0×10^{8}

watts	btu/hr.	3.4129
watts	btu/min.	5.688×10^{-2}
watts	ergs/sec.	1.0×10^{7}
watts	ft.-lbs./min.	4.427×10^{1}
watts	ft.-lbs./sec.	7.378×10^{-1}
watts	horsepower	1.341×10^{-3}
watts	horsepower (metric)	1.36×10^{-3}
watts	kg.-calories/min.	1.433×10^{-2}
watts	kilowatts	1.0×10^{-3}
watts (abs.)	joules/sec.	1.0
watt-hours	btu	3.413
watt-hours	ergs	3.6×10^{10}
watt-hours	foot-lbs.	2.656×10^{3}
watt-hours	gram-calories	8.605×10^{2}
watt-hours	horsepower-hours	1.341×10^{-3}
watt-hours	kilogram-calories	8.605×10^{-1}
watt-hours	kilogram-meters	3.672×10^{2}
watt-hours	kilowatt-hours	1.0×10^{-3}
watt (international)	watt (absolute)	1.000165
webers	maxwells	1.0×10^{8}
webers	kilolines	1.0×10^{5}
webers/sq. meter	lines/sq. in.	6.452×10^{4}
webers/sq.meter	webers/sq. cm.	1.0×10^{-4}
webers/sq. meter	webers/sq. in.	6.452×10^{-4}
weeks	hours	1.68×10^{2}
weeks	minutes	1.008×10^{4}
weeks	seconds	6.048×10^{5}

yards	centimeters	9.144×10^{1}
yards	kilometers	9.144×10^{-4}
yards	meters	9.144×10^{-1}
yards	miles (nautical)	4.934×10^{-4}
yards	miles (statute)	5.682×10^{-4}
yards	millimeters	9.144×10^{2}
years	days (mean solar)	3.65256×10^{2}
years	hours (mean solar)	8.7661×10^{3}

Placer
Problem

PLACER PROBLEM

VEN: Placer property 25 miles from nearest highway via an unimproved gravel road. Nearest town is 20 miles from the turn-off to the property. Six claims located at the 3000 foot elevation in a valley 300 feet wide. Smith Creek flows year round at 5 f^{3}pm. Gravels are estimated to be 5 to 15 feet deep covered by 2 to 4 feet of overburden. Boulder content of the gravel is estimated at 20 %. Pay values are in the lower 2 feet and 1 foot into bedrock and averages 0.025 ounces per yard. Gravels average 3000 lbs. per yard.

Mine rate: 250 yards processed per 8 hour day.
Mine employs 3 men including the owner.
Equipment on site includes: D-8 dozer, 3/4 yd^3 backhoe/excavator trommel screen feeding a 20 foot tri-channel sluice box with 32" wide channels, 6 Kw generator, pickup truck, camp buildings, tool and storage sheds.

PROBLEM ONE

What method(s) would you use to sample this property?

List three advantages and three disadvantages of the chosen method(s).

List what you would do to minimize salting of your samples.

PROBLEM TWO

Using the sampling method chosen in Problem one, calculate the value of each sample.

Calculate the reserves of the property using the block and/or triangle method.

Using IC 9170, calculate the operating costs for this property.

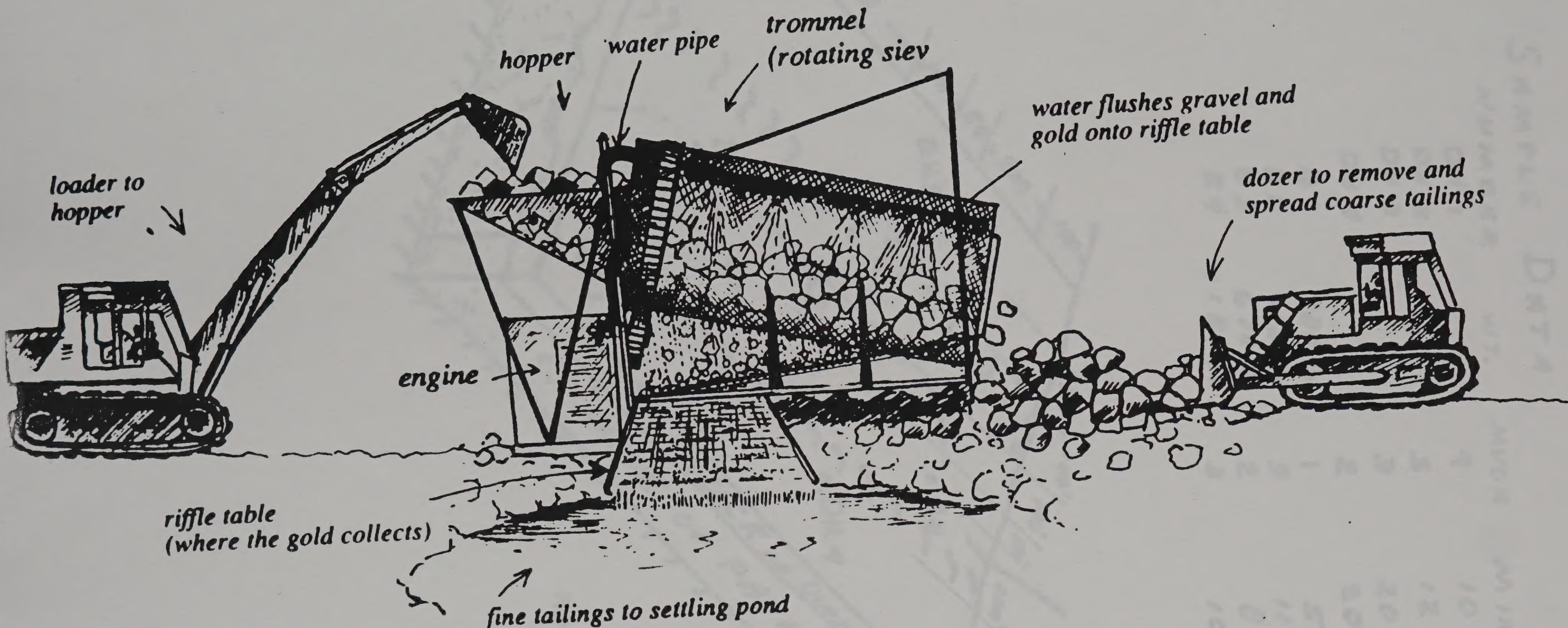
THIS PAGE INTENTIONALLY BLANK

Mining cut.
Cut drainage (flows to ponds).
Overburden.

THIS PAGE INTENTIONALLY BLANK

Public Affairs ...
Ministry ...
New Zealand. ...

A typical alluvial plant extracting gold at Blac... ...er on the We... Coast. The hydraulic excavator or digger feeds gravel into ... rotating screen which sieves the finer material. This finer materi... is passed over a riffle table where the gold is caught in the groov... The line drawing shows larger rocks passing through the rotati... drum sieve (trommel) and being cleared by a bulldozer in the init... stages of land rehabilitation.

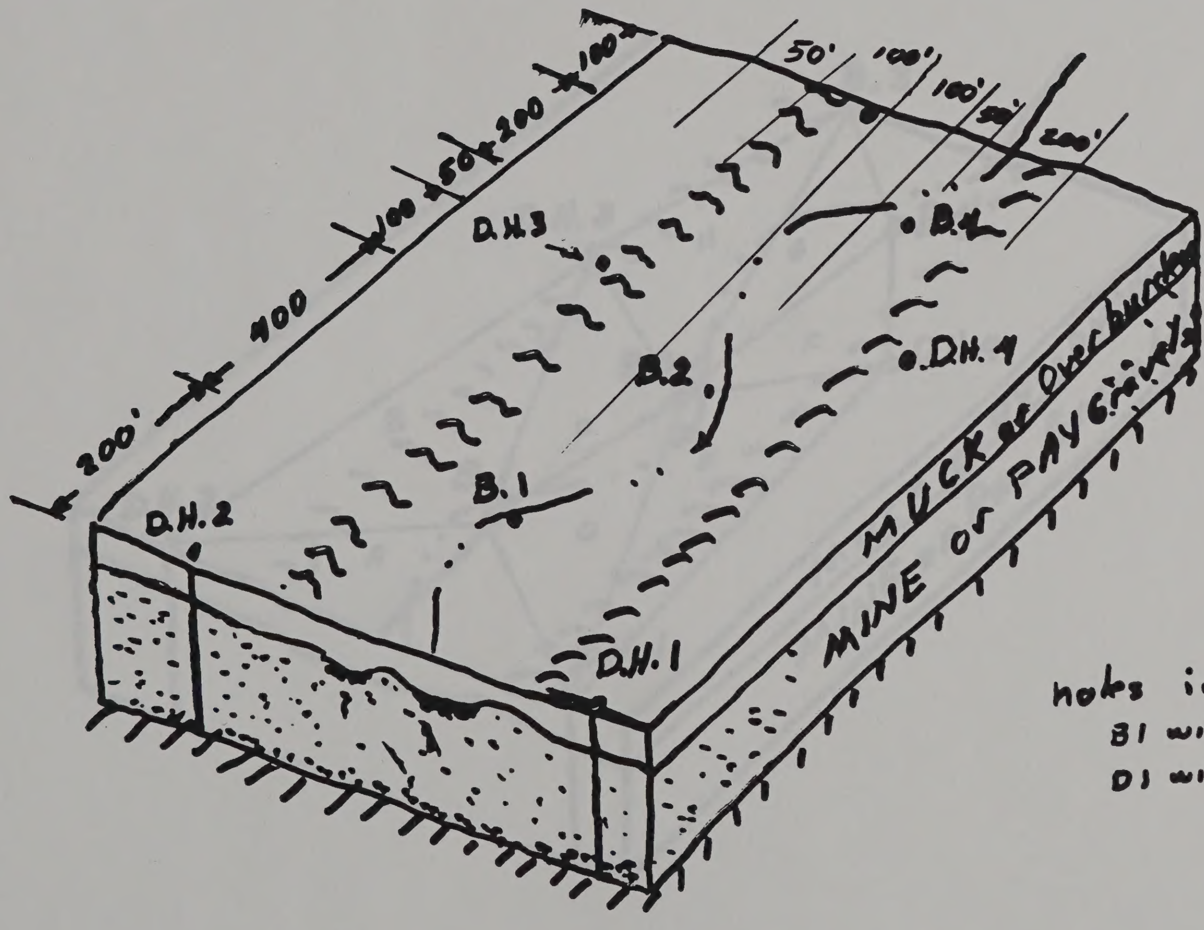


THIS PAGE
INTENTIONALLY
BLANK

SAMPLE DATA

NUMBER	WT.	MUCK	MINE	AU. REC.
DH 1		4	10	15.8 m
DH 2		3	12	31.1 m
DH 3		3	20	18.2 m
DH 4		2	20	48.3 m
B 1	780 lb	1	5	79.9 m
B 2	912 lb	3	11	171.5 m
B 3	670 lb	2	8	91.5 m
B 4	1180 lb	3	10	293.0 m

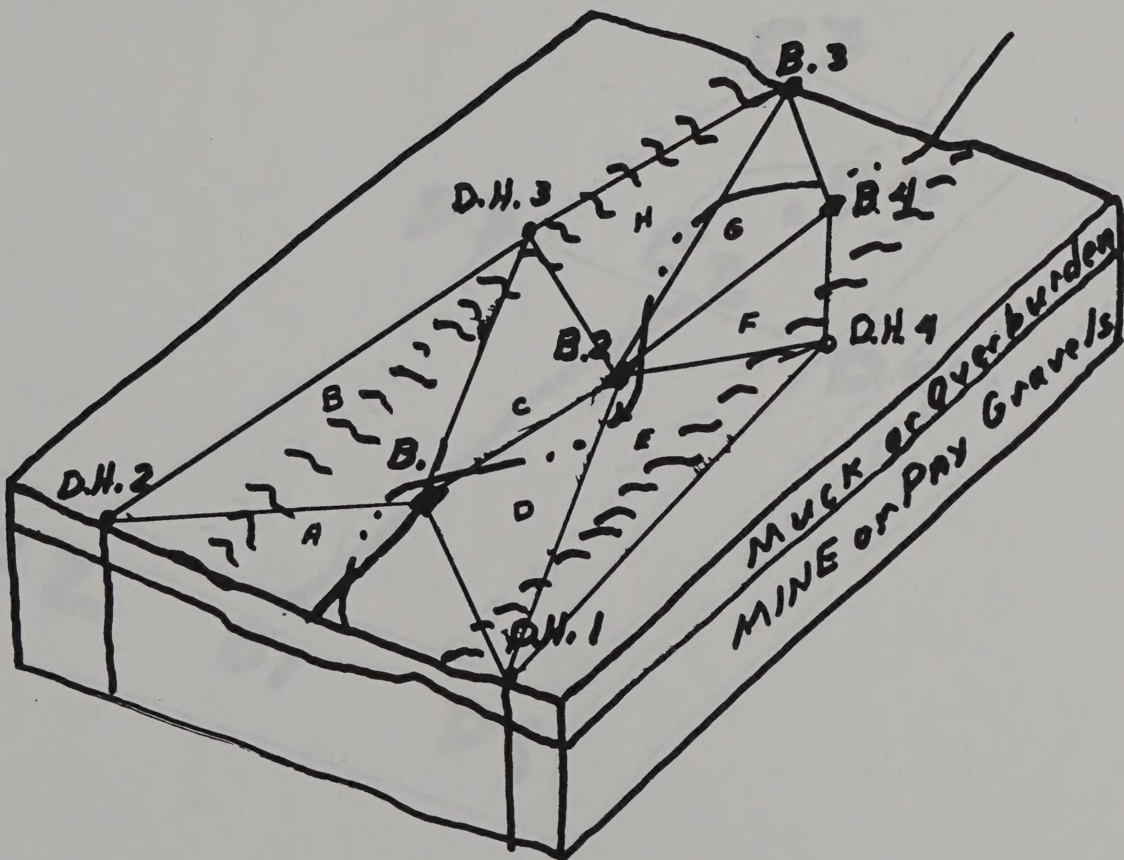
1 yd³ = 3000 lb



holes in line:
B1 with B3
D1 WITH D4

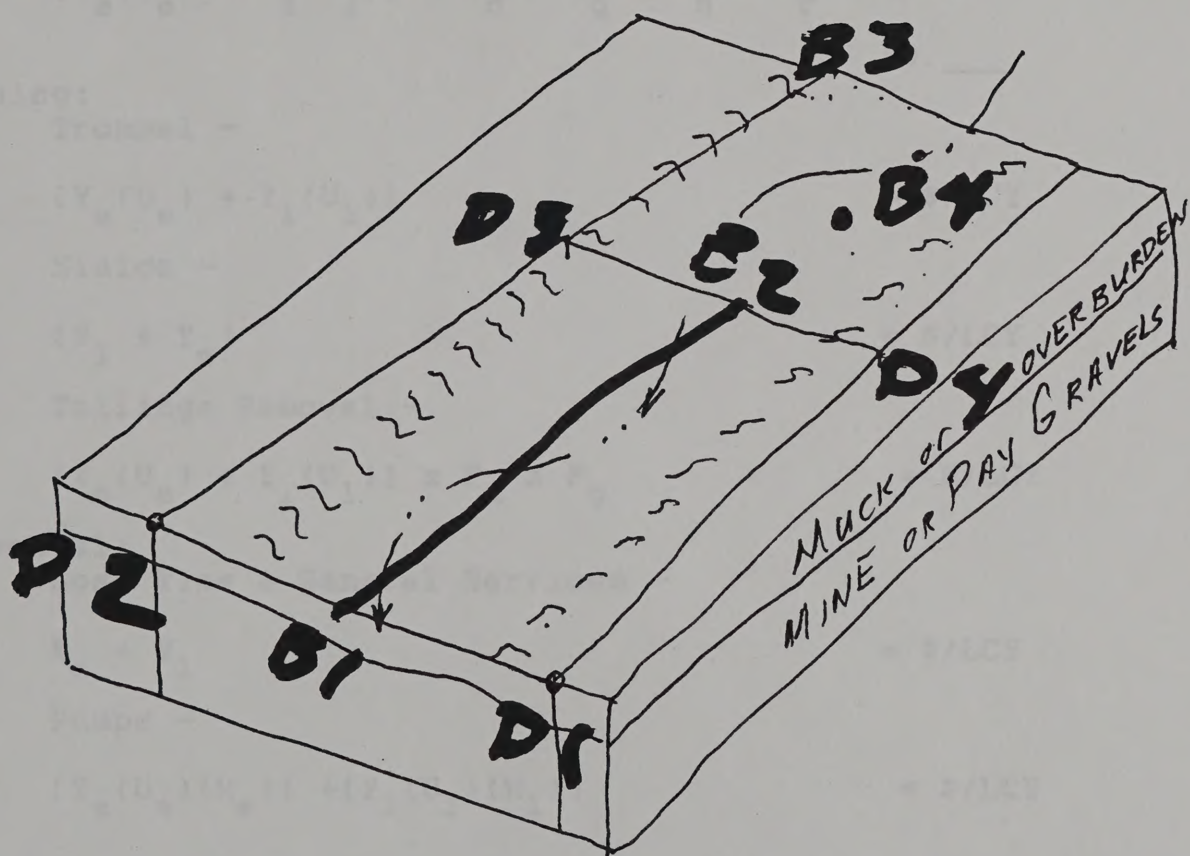
THIS PAGE
INTENTIONALLY
BLANK

RESERVE ESTIMATE
TRIANGLE METHOD



THIS PAGE
INTENTIONALLY
BLANK

BLOCK METHOD



THIS PAGE
INTENTIONALLY
BLANK

COST ESTIMATE

from USBM IC 9170

Overburden Removal:

Bulldozers-

$[Y_e(U_e) + Y_1(U_1)] \times F_d \times F_g \times F_h \times F_r$ = \$/LCY

\$0.

Mining :

Backhoes -

$[Y_e(U_e) + Y_1(U_1)] \times F_d \times F_g \times F_h \times F_r$ = \$/LCY

\$0.___

Processing:

Trommel -

$[Y_e(U_e) + Y_1(U_1)]$ = \$/LCY

Sluice -

$[Y_1 + Y_e]$ = \$/LCY

Tailings Removal -

$[Y_e(U_e) + Y_1(U_1)] \times F_d \times F_g$ = \$/LCY

Supplemental:

Lost Time & General Services -

$Y_e + Y_1$ = \$/LCY

Pumps -

$[Y_e(U_e)(H_e)] + [Y_1(U_1)(H_1)]$ = \$/LCY

THIS PAGE
INTENTIONALLY
BLANK

Bob Lewis:
Estimating Resources

MINE DESCRIPTION

Prob 2

BCY ORE PER DAY PROCESSED=	250	HRS/YEAR=	720
DAYS PER YEAR=	90	LCY ORE/YR=	29250
HOURS PER DAY=	8	LCY ORE/HR=	40.63
SWELL FACTOR (%)=	30	LCY OVBD/HR	35.75
% GRAVEL SCREENED BEFORE PLANT	100	OVBD+ORE/HR	76.38
TOTAL BCY OVBD REM'VD FROM MINE	19400		
BCY OVBD PER DAY REMOVED=	220		

--- CAPITAL COSTS ---

PREPRODUCTION CAP COST	$3,284
MINE CAP COST	$77,178
MILL CAP COST	$62,185
SUPPLEMENTAL	$57,992
	===========
TOTAL	$200,639

--- OPERATING COSTS ---

OVERBURDEN REMOVAL	$0.67	
MINING	$0.89	
PROCESSING	$0.18	
TAILINGS REMOVAL	$0.51	
SUPPLEMENTAL	$0.89	
	===========	
TOTAL	$3.14	PER YARD

THIS PAGE INTENTIONALLY BLANK

ter of the deposit and for which there are few, if any, samples or measurements. The estimates are based on an assumed continuity or repetition, of which there is geologic evidence; this evidence may include comparison with deposits of similar type. Bodies that are completely concealed may be included if there is specific geologic evidence of their presence. Estimates of inferred reserves or resources should include a statement of the specific limits within which the inferred material may lie.

Identified-Subeconomic.—Resources that are not Reserves, but may become so as a result of changes in economic and legal conditions.

Paramarginal.—The portion of Subeconomic Resources that (1) borders on being economically producible or (2) is not commercially available solely because of legal or political circumstances.

Submarginal.—The portion of Subeconomic Resources which would require a substantially higher price (more than 1.5 times the price at the time of determination) or a major cost-reducing advance in technology.

Hypothetical resources.—Undiscovered resources that may reasonably be expected to exist in a known mining district under known geologic conditions. Exploration that confirms their existence and reveals quantity and quality will permit their reclassification as a Reserve or Identified-Subeconomic resource.

Speculative resources.—Undisovered resources that may occur either in known types of deposits in a favorable geologic setting where no discoveries have been made, or in as yet unknown types of deposits that remain to be recognized. Exploration that confirms their existence and reveals quantity and quality will permit their reclassification as Reserves or Identified-Subeconomic resources.

AREAS OF RESPONSIBILITY AND OPERATIONAL PROCEDURES

U.S. Bureau of Mines.—The Bureau appraises, analyzes, and publishes reserve estimates from base data supplied by the mineral and energy materials industry, the U.S. Geological Survey, and other governmental agencies. The Bureau judges commodity recoverability on existing economic and legal factors.

U.S. Geological Survey.—The Survey appraises, analyzes, and publishes estimates of Total Resources. It reports such measurable parameters of significance to resource evaluation as location, quality, quantity, and situation of Identified resources.

Annual Resource Summation.—The U.S. Bureau of Mines and U.S. Geological Survey will confer and agree annually on estimates in all of the resource categories defined above. These data will be in Bureau or Survey publications and will be available for inclusion in the Secretary's Annual Report required by the Mining and Minerals Policy Act of 1970.

Ad Hoc Joint Conferences.—The Directors will convene ad hoc joint work groups to resolve problems in the resource area.

To serve these planning purposes Total Resoures are classified both in terms of economic feasibility and of the degree of geologic assurance. The factors involved are incorporated in figure 1 to provide a graphic classification of Total Resources.

General guides for the use of this classification system are as follows:

1. Resource categories and definitions in the classification, as specified in the glossary, should be applicable to all naturally occurring concentrations of metals, nonmetals, and fossil fuels. The categories may be subdivided for special purposes.

2. Definitions may be amplified, where necessary, to make them more precise and conformable with accepted usage for particular commodities or types of resource evaluations.

3. Quantities and qualities may be expressed in a variety of terms and units to suit different purposes, but must be clearly stated and defined.

GLOSSARY OF RESOURCE TERMS

Resource.—A concentration of naturally occurring solid, liquid, or gaseous materials in or on the Earth's crust in such form

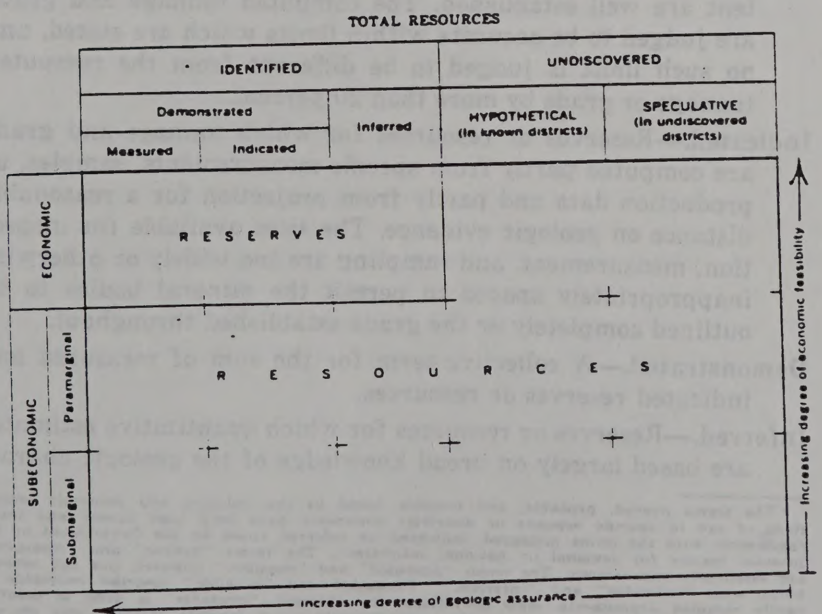
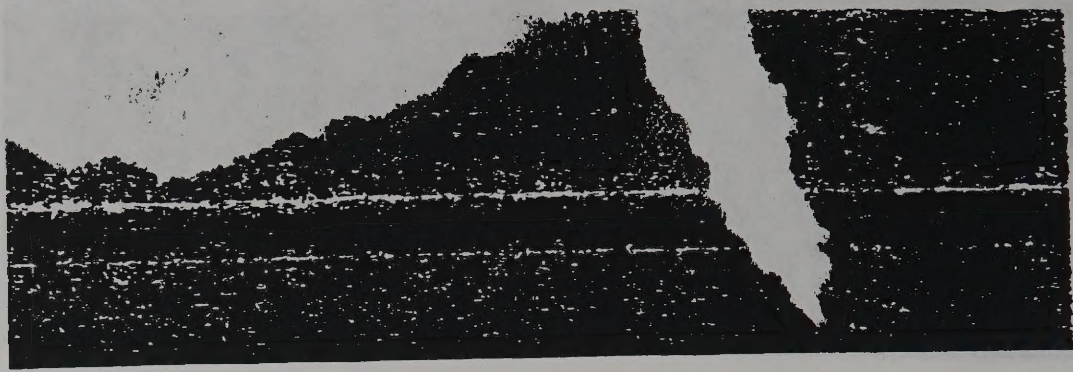


FIGURE 1.—Classification of mineral resources.



that economic extraction of a commodity is currently or potentially feasible.

Identified resources.—Specific bodies of mineral-bearing material whose location, quality, and quantity are known from geologic evidence supported by engineering measurements with respect to the demonstrated category.

Undiscovered resources.—Unspecified bodies of mineral-bearing material surmised to exist on the basis of broad geologic knowledge and theory.

Reserve.—That portion of the identified resource from which a usable mineral and energy commodity can be economically and legally extracted at the time of determination. The term *ore* is used for reserves of some minerals.

The following definitions for measured, indicated, and inferred are applicable to both the Reserve and Identified-Subeconomic resource components.[1]

Measured.—Reserves or resources for which tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are spaced so closely and the geologic character is so well defined that size, shape, and mineral content are well established. The computed tonnage and grade are judged to be accurate within limits which are stated, and no such limit is judged to be different from the computed tonnage or grade by more than 20 percent.

Indicated.—Reserves or resources for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement, and sampling are too widely or otherwise inappropriately spaced to permit the mineral bodies to be outlined completely or the grade established throughout.

Demonstrated.—A collective term for the sum of measured and indicated reserves or resources.

Inferred.—Reserves or resources for which quantitative estimates are based largely on broad knowledge of the geologic charac-

[1] The terms proved, probable, and possible (used by the industry and economic evaluations of ore in specific deposits or districts) commonly have been used loosely and interchangeably with the terms measured, indicated, or inferred (used by the Department of the Interior mainly for regional or national estimates). The terms "proved" and "measured" are essentially synonymous. The terms "probable" and "possible," however, are not synonymous with "indicated" and "inferred." "Probable" and "possible" describe estimates of partly sampled deposits—in some definitions, for example, "probable" is used to describe deposits sampled on two or three sides, and "possible" for deposits sampled only on one side; in the Bureau-Survey definitions, both would be described by the term "indicated."

[redacted]

CONTENTS

ILLUSTRATION

MINERAL RESOURCE CLASSIFICATION SYSTEMS OF THE U.S. BUREAU OF MINES AND U.S. GEOLOGICAL SURVEY

PRINCIPLES OF THE MINERAL RESOURCES CLASSIFICATION SYSTEM OF THE U.S. BUREAU OF MINES AND U.S. GEOLOGICAL SURVEY

GENERAL DEFINITION OF MINERAL AND ENERGY RESOURCES

The dictionary definition of resource "something in reserve or ready if needed" has been extended for mineral and energy resources to comprise all materials surmised to exist having present or future values. In geologic terms a mineral or energy resource is a concentration of naturally occurring solid, liquid, or gaseous materials in or on the Earth's crust in such form that economic extraction of a commodity is currently or potentially feasible. Material classified as a reserve is that portion of an identified resource producible at a profit at the time of classification.

Total Resources are materials that have present or future value and comprise identified or known materials plus those not yet identified, but which on the basis of geologic evidence are presumed to exist.

PHILOSOPHIC BASIS FOR A RESOURCE CLASSIFICATION

Public attention usually is focused on current economic availability of mineral or energy materials (reserves). Long-term public and commercial planning, however, must be based on the probability of geologic identification of resources in as yet undiscovered deposits and of technologic development of economic extraction processes for presently unworkable deposits. Thus, all the components of Total Resources must be continuously reassessed in the light of new geologic knowledge, of progress in science, and of shifts in economic and political conditions.

Another requirement of long-term planning is the weighing of total or multi-commodity resource availability against a particular need. To achieve this the general classification system must be uniformly applicable to all commodities so that data for alternate or substitute commodities can be compared.

Principles of the Mineral Resource Classification System of the U.S. Bureau of Mines and U.S. Geological Survey

MINERAL RESOURCE CLASSIFICATION SYSTEMS OF THE U.S. BUREAU OF MINES AND U.S. GEOLOGICAL SURVEY

GEOLOGICAL SURVEY BULLETIN 1450-A

A report published jointly by the U.S. Bureau of Mines and U.S. Geological Survey

Definitions of mineral resource classification terms used by the U.S. Bureau of Mines and U.S. Geological Survey

UNITED STATES GOVERNMENT PRINTING OFFICE, WASHINGTON : 1976

FOREWORD

In order to use mineral resource terms with precision and common understanding and to compare resource data effectively, a joint U.S. Bureau of Mines and U.S. Geological Survey work group developed a standardized, definitive, broadly applicable classification system to derive uniform, coordinated resource estimates. This report presents the results of the work group that developed the basic terms of mineral resource classification. Other chapters in this series will present classification terms for specific commodities.

Thomas V. Falkie
Director, Bureau of Mines

V. E. McKelvey
Director, Geological Survey

United States Department of the Interior

OFFICE OF HEARINGS AND APPEALS
Interior Board of Land Appeals
4015 Wilson Boulevard
Arlington, Virginia 22203

IN REPLY REFER TO

VANDERBILT GOLD CORP.

IBLA 92-322 Decided April 19, 1993

Appeal from a decision of the Oregon State Office, Bureau of Land Management, rejecting hardrock prospecting permit application OR 45170 (WASH).

Affirmed as modified.

1. Mineral Lands: Prospecting Permits

The Secretary has discretionary authority to issue mineral prospecting permits when prospecting or exploratory work is necessary to determine the existence or workability of a particular hardrock mineral deposit. It is not appropriate to issue a prospecting permit if there is sufficient data regarding the quality and quantity of a deposit to conclude that only an increase in the price of the commodity would render a deposit workable.

2. Mineral Lands: Prospecting Permits

BLM may reject a prospecting permit if further prospecting or exploratory work would not disclose the existence or workability of a deposit of hardrock mineral. A deposit is considered workable when the value of the commodity is greater than the cost of extracting it. A workability determination is made by examining only those factors directly related to production of the mineral.

3. Administrative Procedure: Hearings--Hearings--Rules of Practice: Appeals: Hearings--Rules of Practice: Hearings

Although the Board has discretionary authority to order a hearing before an Administrative Law Judge pursuant to 43 CFR 4.415, a hearing is necessary only when there is a material issue of fact requiring resolution through the introduction of testimony and other evidence.

4. Evidence: Burden of Proof—Mineral Lands: Prospecting Permits

When considering what evidentiary burden should be placed upon BLM in an appeal from a rejection of a prospecting permit application, it is proper to weigh the cost of that burden against the nature of the appellant's interest and the risk that an appellant would be improperly deprived of that interest if the greater burden were not placed on BLM. A prospecting permit applicant holds an expectancy and not an interest in the land and BLM is not required to sustain its workability determinations with the same quantum of evidence needed to sustain a discovery determination under the 1872 Mining Law.

APPEARANCES: Thomas P. Erwin, Esq., Reno, Nevada, for appellant and intervenor, Teck Resources, Inc.; Donald P. Lawton, Esq., Assistant Regional Solicitor, Portland, Oregon, for the Bureau of Land Management.

OPINION BY ADMINISTRATIVE JUDGE MULLEN

Vanderbilt Gold Corporation (Vanderbilt) has appealed from the February 21, 1992, decision of the Oregon State Office, Bureau of Land Management (BLM), rejecting hardrock prospecting permit application OR 45170 (WASH). Vanderbilt had filed an application for a permit to prospect for copper, molybdenum, gold, silver, and associated minerals on 897.9 acres of acquired lands within portions of secs. 7, 8, 9, 16, 17, 18, 19, and 20, T 10 N., R. 6 E., Willamette Meridian, Washington. This land also lies within the Gifford Pinchot National Forest. By order dated May 26, 1992, Teck Resources, Inc. (Teck), was allowed to intervene. 1/

[1] The Secretary has been granted discretionary authority to issue mineral prospecting permits and leases. Prospecting permit issuance is authorized only if "prospecting or exploratory work is necessary to determine the existence or workability of a particular hardrock mineral(s)." 43 CFR 3562.1. However, this regulation does not mandate permit issuance whenever available evidence does not indicate that a known deposit is workable. For example, if a known deposit cannot be considered workable because the value of the mineral in place is less than the cost of extracting it, and further prospecting or exploratory work would not result in a significant change in its size or grade, a prospecting permit would not be warranted. Said another way, it would not be appropriate to issue a prospecting permit when sufficient data exists regarding the quality and quantity of a deposit to conclude that only an increase in the price of the commodity would render the deposit workable. Prospecting permits are

---

1/ Teck is the lessee of fee lands adjacent to those described in the permit application from Vanderbilt, and Vanderbilt has granted Teck the "right to obtain an assignment of the prospecting permit upon issuance."

issued to encourage genuine prospecting, and are not a proper vehicle for holding a known subeconomic deposit in anticipation of more favorable economic conditions.

[2] When prospecting or exploratory work is unnecessary to determine the existence or workability of a valuable deposit of a particular hardrock mineral, the land can only be leased through competitive sale. 43 CFR 3564.1. BLM rejected Vanderbilt's application because the deposit was deemed to be "workable." In its decision the term "workable deposit" was defined in the following manner:

> A deposit is determined to be "workable" when the value of the commodity is greater than the cost of extracting. Workability is only concerned with the economics of intrinsic factors involved in the development of the deposit and not extrinsic factors such as transportation, markets, etc., involved in the actual production and marketing of the commodity. See Elizabeth B. Archer et. al.[,] 102 IBLA 308 (1988), United States Gypsum Co.[,] 121 IBLA 174 (1991), and American Gilsonite Co.[,] 111 IBLA 1 (1989).

BLM's decision was based on a formal mineral report (BLM Report) prepared by Denny R. Seymour, a BLM mining engineer, and Diane Groody, a BLM geologist. The authors describe the applied-for land in the following manner:

> The subject lands are located in the northern portion of the Mt. St. Helens Mining District along the Green River and on the south slope of Goat Mountain one-half mile to the west of Ryan Lake and eleven and [one] half miles to the northeast of Mt. St. Helens. Three of the patents were issued prior to 1911 and two patents were issued in 1982 to the Duval Corporation based on their discovery and exploration of the Margaret Deposit during the 1970's. The Margaret Deposit is a major porphyry copper deposit suitable for mining as an open pit. Duval estimated the deposit to contain 577 million tons of 0.36% copper (Cu), 0.011% molybdenum (Mo), 0.007 oz/ton gold (Au), and 0.046 oz/ton silver (Ag) at a 0.33% Cu equivalent cutoff.

(BLM Report at 1).

During the 1970's, Duval Corporation was a major copper producer operating some of the lowest grade copper/molybdenum open pit mines in the United States. Both the U.S. Forest Service and BLM concluded that Duval had discovered a valuable mineral deposit and issued a mineral patent for a portion of the Margaret Deposit (Affidavit of Denny R. Seymour at 16). Duval's parent company, Pennzoil, elected to divest itself of hard mineral mining operations in 1984, and transferred Duval's patented and unpatented claims in the Mt. St. Helens Mining District to the Trust for Public Land (Trust). The Trust abandoned the unpatented claims in 1985 and conveyed the patented claims to the Forest Service (BLM Report at 5).

Using data from Duval's mine patent application files, the author(s) created a computer model of the deposit's recovered copper equivalent grade. The equivalent recovered grade was based on the estimated long-term metal prices of $1.00/lb. Cu, $3.50/lb. Mo, $400.00/oz Au, and $6.00/oz Ag (Spring 1990) and metal recoveries of 90% (Cu), 80% (Mo and Ag), and 50% (Au). To evaluate the economics of mining the higher grade portions of the deposit, three different "higher" grade open pits were delineated. In addition, Duval generated two different resource assessments for a 577 million ton reserve and a higher grade reserve containing 244 million tons. From these five development scenario's [sic] economic assessments were generated assuming various copper prices and mining rates for each scenario. The economic analyses were generated using the U.S. Bureau of Mines cost estimating procedures for unit mining costs, and <u>Mining Cost Service</u>'s (Western Mine Engineering) cost models for unit milling costs and for capital costs for mine and mill. The resulting analyses indicate that at current long term market prices (1.10/lb. Cu) none is likely to generate a competitive rate of return on invested capital at copper prices of about $1.25/lb. the pre-tax rates of return approach 15%. [sic] Duval's two development scenario's [sic] indicate that revenues exceed expected costs at current long term market prices of $1.10/lb. Cu, $3.15/lb. Mo, $393.00/oz Au, and $5.28/oz Ag (Fall 1991).

(BLM Report at 1).

The examining team's conclusion, set out in the BLM Report, was:

Based on the data available to the author(s) it is concluded that the lands applied for under prospecting permit application, OR 45170 (Wash), <u>do not</u> require prospecting and/or exploratory work to determine the existence and workability of the hardrock minerals which have been applied for. This conclusion is based on the economic models developed for this report for the Margaret deposit and the similarity of its reserve estimates to several major operating open pit mines with a similar range of operating conditions, locations, and geologic environments (Table #1 & Table #2). The "net dollars", shown in Table #2, is the sum of expected revenue less all project costs including pre-production, mine, mill, and smelter operating and initial capital costs. The test of workability is met for a given development scenario when a positive value occurs in the "Net" column of Table #2 for a copper price of $1.10/lb. or less. None of the scenarios developed for this report denoted as BLMPPA (Bureau of Land Management Prospecting Permit Application) proved viable at price levels less than or equal to $1.30/lb.. However, the scenarios which were developed based on Duval's reserve estimates generate positive Net values at $1.10/lb. Cu for all projected mining rates except one. The return on capital (ROC) is the rate of return the scenario would generate on the initial capital investment with no taxes.

> It appears that price levels of at least $1.25/lb. to $1.30/lb. are needed to justify commercial operations at current state of knowledge (or at least 15% ROC). At long term metal prices each percentage unit of molybdenum is equivalent to 2.86 percentage units of copper or approximately 1 to 3 (x% Mo = 3x% Cu). All of the mines in Table #1 have reserve grades comparable to Duval's reserve estimates. [Emphasis in original.]

(BLM Report at 2).

In its statement of reasons (SOR), Vanderbilt contends that the data used by BLM does not support a finding that the minerals on the land are workable, and that further exploration and prospecting is necessary. A mineral report based on existing data was prepared by Steffen, Robertson and Kirsten, Inc. (SRK). The SRK report was submitted in support of Vanderbilt's statement of reasons and its contention that existing data do not provide a reliable basis for classifying any mineral deposit in the applied-for area as workable.

Vanderbilt has requested a hearing before an Administrative Law Judge to permit it to offer evidence and cross-examine BLM employees. BLM opposes this request, contending that its study was based on the same factual data as that used by Vanderbilt, reducing the dispute to a difference of opinion among experts. BLM refers to our holding in American Gilsonite Co., 111 IBLA 1, 96 I.D. 408 (1989), that a mere difference of opinion among experts will not suffice to reverse a reasoned opinion rendered by the Secretary's technical staff.

Hearings are not a normal feature in the adjudication of cases such as this. E.g., United States Gypsum Co., 121 IBLA 174 (1991); American Gilsonite Co., 111 IBLA 1, 96 I.D. 408 (1989); A. J. Maurer, Jr., 106 IBLA 308 (1989); Earth Sciences, Inc., 80 IBLA 28 (1984); Christian F. Murer, 75 IBLA 232 (1983); John D. Archer, 75 IBLA 128 (1983); J. R. Simplot Co., 58 IBLA 305 (1981); Christian F. Murer, 57 IBLA 333 (1981); Philip Shaiman, 25 IBLA 271 (1976); William F. Martin, 24 IBLA 271 (1976); Powhattan Mining Co., 10 IBLA 308 (1973); William J. Colman, 9 IBLA 15 (1973); Lloyd K. Johnson, 8 IBLA 73 (1972); Clear Creek Inn Corp., 7 IBLA 200, 79 I.D. 571 (1972); J. D. Archer, 4 IBLA 323 (1972); J. D. Archer, 1 IBLA 26, 77 I.D. 124 (1970). The first hearing held in this type of case that we find was noted in James C. Goodwin, 9 IBLA 139, 143, 80 I.D. 7, 9 (1973). A hearing was ordered in Elizabeth B. Archer, 82 IBLA 14 (1984), and the result of that hearing was reviewed in Elizabeth B. Archer, 102 IBLA 308 (1988). 2/

2/ When a mining claim is challenged for lack of discovery of valuable mineral (General Mining Law, 30 U.S.C. § 22 (1988)), a contest hearing is held pursuant to 5 U.S.C. § 554 (1988). The mining claim constitutes a property interest which cannot be invalidated on the basis of a disputed issue of fact without notice and an opportunity for a hearing. United States v. O'Leary, 63 I.D. 341 (1956). A hearing is not usually required

[3] The Board has discretionary authority to order a hearing before an Administrative Law Judge pursuant to 43 CFR 4.415, but will normally order a hearing only when it finds a material issue of fact that can only be resolved through the introduction of testimony and other evidence not readily obtainable through the ordinary appeal procedure. See United States v. Consolidated Mines & Smelting Co., 455 F.2d 432, 453 (9th Cir. 1971); Ben Cohen (Judicial Remand), 103 IBLA 316, 321, aff'd sub nom., Sahni v. Watt, Civ. No. S-83-96-HDM (D. Nev. Jan. 17, 1990), aff'd (Jan. 14, 1991); KernCo Drilling Co., 71 IBLA 53, 56 (1983). If no oral testimony is required and an appeal can be resolved relying on documentary submissions, a request for a hearing is properly denied. See R. A. Mikelson, 26 IBLA 1 (1976). In Ben Cohen, supra, we stated: "[T]o determine whether a material issue of fact exists, the Board first examines the legal principles which govern its consideration of an appeal on the basis of facts which are not in dispute. E.g., KernCo Drilling Co., [supra,]."

Vanderbilt and BLM present their arguments as if prospecting permit denial must be based on a finding that a "reserve" exists on the land subject to the permit application. Concluding that the deposit is workable, BLM's Report states:

> This conclusion is based on the economic models developed for this report for the Margaret deposit and the similarity of its reserve estimates to several major operating open pit mines with a similar range of operating conditions, locations, and geologic environments (Table #1 & Table #2). The "net dollars", shown in Table #2, is the sum of expected revenue less all project costs including pre-production, mine, mill, and smelter operating and initial capital costs. The test of workability is met for a given development scenario when a positive value occurs in the "Net" column of Table #2 for a copper price of $1.10/lb. or less.

(BLM Report at 2).

Appellant also equates the workability test to a definition of a "reserve," as that term is defined by the Society for Mining, Metallurgy, and Exploration, Inc. (SME or Society) in A Guide for Reporting Exploration Information, Resources and Reserves, 43 Mining Engineering 379-84 (April 1991) (Exhibit C to the SRK report) (SRK Report at 21). It also bases its detailed objections to the sufficiency of BLM's data and analysis on these

---

fn. 2 (continued)
for a "workability" determination, even though it involves similar issues, because a prospecting permit applicant gains no property right when filing an application, and there is no basis for asserting a due process requirement for a hearing and an opportunity to cross-examine. Christian F. Murer, 75 IBLA 232 (1983); John D. Archer, 75 IBLA 128 (1983); J. R. Simplot Co., 58 IBLA 305 (1981); Christian F. Murer, 57 IBLA 333 (1981); William F. Martin, 24 IBLA 271 (1976).

criteria, and consideration of appellant's arguments is facilitated by a brief discussion of mineral classification.

There is no firmly established or codified set of definitions of terms (standards) applied by the industry when classifying a mineral deposit. The term "reserves" is found to have differing meanings in different contexts. Terms with separate and distinct meaning are often used interchangeably and in inappropriate contexts. This often results in confusion regarding what was intended when the viability of a mineral property is discussed.

We find no decision setting out the relationship between statutory and regulatory standards such as "workability" and the concept of a "reserve" with the degree of precision Vanderbilt would apply. The SME committee report reflects an ongoing professional effort to develop more accurate and acceptable definitions. Before attempting to determine whether "reserve" and a "workable deposit" can be considered synonymous, we must define the term "reserve."

The Department has used similar definitions found in Principles of the Mineral Resources Classification System, Geological Survey Bulletin 1450-A (Bulletin) when classifying mineral deposits under other statutes. 3/ See, e.g., United States v. Feezor, 74 IBLA 56, 83-85, 90 I.D. 262, 277-78 (discussing relationship of reserve classification to issue of whether there was a discovery of a valuable mineral deposit under the mining laws). The Geological Survey (Survey) definitions define a "resource" as "a concentration of naturally occurring solid, liquid, or gaseous materials in or on the earth's crust in such form that economic extraction of a commodity is currently or potentially feasible." Bulletin at A2-A3. Two resource categories are given:

> **Identified resources.**—Specific bodies of mineral-bearing material whose location, quality, and quantity are known from geologic evidence supported by engineering measurements with respect to the demonstrated category.
>
> **Undiscovered resources.**—Unspecified bodies of mineral-bearing material surmised to exist on the basis of broad geologic knowledge and theory. [Bold in original.]

Id. at A3. 4/ The Survey description of a resource gives no suggestion that any given deposit will have economic value. The concept of economic value becomes applicable when the term "reserve" is employed. A "reserve" is "[t]hat portion of the identified resource from which a usable mineral and energy commodity can be economically and legally extracted at the time of determination." 5/ Vanderbilt contends that the value of the known copper

---

3/ The Survey definitions were used extensively prior to 1991, when the more precise SME definitions were published.
4/ Undiscovered resources are further divided into hypothetical resources and speculative resources. Bulletin at A4.
5/ Survey lists the following categories of identified resources that do not qualify as "reserves":

eposit is too low to permit economic extraction, the deposit cannot be called a reserve, and it is therefore not workable.

Using the approach adopted by both Survey and SME, deposits are classified on the basis of an increasing degree of geologic assurance and on the basis of an increasing degree of certainty regarding economic feasibility. Survey developed the following definitions, applicable to both identified-subeconomic resources and reserves:

Measured.—Reserves or resources for which tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are spaced so closely and the geologic character is so well defined that size, shape, and mineral content are well established. The computed tonnage and grade are judged to be accurate within limits which are stated, and no such limit is judged to be different from the computed tonnage or grade by more than 20 percent.

Indicated.—Reserves or resources for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement, and sampling are too widely or otherwise inappropriately spaced to permit the mineral bodies to be outlined completely or the grade established throughout.

Demonstrated.—A collective term for the sum of measured and indicated reserves or resources.

Inferred.—Reserves or resources for which quantitative estimates are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. The estimates are based on an assumed continuity or repetition, of which there is geologic evidence; this evidence may include comparison with deposits of similar type. Bodies that are completely concealed may be included if there is specific geologic evidence of their presence. Estimates

---

fn. 5 (continued)

"Identified-Subeconomic.—Resources that are not Reserves, but may become so as a result of changes in economic and legal conditions.

"Paramarginal.—The portion of Subeconomic Resources that (1) borders on being economically producible or (2) is not commercially available solely because of legal or political circumstances.

"Submarginal.—The portion of Subeconomic Resources which would require a substantially higher price (more than 1.5 times the price at the time of determination) or a major cost-reducing advance in technology."

Id. at 4 (bold in original).

> of inferred reserves or resources should include a statement of the specific limits within which the inferred material may lie. [Bold in original.]

Bulletin at A3-A4.

It is not surprising that the SME terminology is similar to that previously developed by Survey. A major difference results from SME's having adopted "a sequential relationship between exploration information, resources and reserves." 43 Mining Engineering 379 (April 1991). Thus, SME begins its classification by examining the available physical information about a mineral deposit. The Society describes exploration information as: "[A]ctivities designed to locate economic deposits and to establish the size, composition, shape and grade of these deposits. Exploration methods include geological, geochemical, and geophysical surveys, drill holes, trial pits and surface and underground openings." Id. at 379-80. SME then offers the following definition of "resource:"

> A concentration of naturally occurring solid, liquid or gaseous material in or on the Earth's crust in such form and amount that economic extraction of a commodity from the concentration is currently or potentially feasible. Location, grade, quality, and quantity are known or estimated from specific geological evidence. To reflect varying degrees of geological certainty, resources can be subdivided into measured, indicated, and inferred.

Id. The first sentence of the SME definition is the same as that developed by Survey. Survey included undiscovered "resources." SME excludes "undiscovered resources" in the last two sentences of its definition, characterizing that classification as "used by public planning agencies [but] not appropriate for use in commercial ventures." Id. at 379. Thus, the SME term "resource" is analogous to the Survey term "identified resource."

Survey and Society both divide resources into three classes: measured, indicated, and inferred. The SME defines these terms as:

> **Measured.** Quantity is computed from dimensions revealed in outcrops, trenches, workings or drill holes; grade and (or) quality are computed from the result of detailed sampling. The sites for inspection, sampling and measurement are spaced so closely and the geological character is so well defined that size, shape, depth and mineral content of the resource are well established.
>
> **Indicated.** Quantity and grade and (or) quality are computed from information similar to that used for measured resources, but the sites for inspection, sampling, and measurements are farther apart or are otherwise less adequately spaced. The degree of

> assurance, although lower than that for measured resources, is high enough to assume geological continuity between points of observation.
>
> **Inferred.** Estimates are based on geological evidence and assumed continuity in which there is less confidence than for measured and (or) indicated resources. Inferred resources may or may not be supported by samples or measurements but the inference must be supported by reasonable geo-scientific (geological geochemical, geophysical, or other) data. [Bold in original.]

Id. The SME definition of "measured" closely follows that of Survey, but Survey adds: "[T]he computed tonnage and grade are judged to be accurate within limits which are stated, and no such limit is judged to be different from the computed tonnage or grade by more than 20 percent."

The definitions of the term "indicated" differ in one respect worth noting. The Survey definition would allow a resource to be categorized as indicated when "the sites available for inspection, measurement, and sampling are too widely or otherwise inappropriately spaced to permit the mineral bodies to be outlined completely or the grade established throughout." SME allows that the points for inspection, sampling, and measurements are less adequately spaced than is required for a measured resource, but states that: "[T]he degree of assurance although lower than that for measured resources, is high enough to assume geological continuity between points of observation."

SME's definition of "inferred" is similar to, but may be considered more restrictive than Survey's. SME omits, for example, the following sentence that appears in Survey's definition: "[B]odies that are completely concealed may be included if there is specific geologic evidence of their presence." However, the language in the Survey's definition requiring "specific geologic evidence" of completely concealed bodies renders that definition very close to the SME definition.

SME's definition of "reserve" is more detailed than Survey's:

> Reserve. A reserve is that part of the resource that meets minimum physical and chemical criteria related to the specified mining and production practices, including those for grade, quality, thickness and depth; and can be reasonably assumed to be economically and legally extracted or produced at the time of determination. The feasibility of the specified mining and production practices must have been demonstrated or can be reasonably assumed on the basis of tests and measurements. The term reserves need not signify that extraction facilities are in place and operative.
>
> The term economics implies that profitable extraction or production under defined investment assumptions has been established or analytically demonstrated. The assumptions made must

> be reasonable including assumptions concerning the prices and costs that will prevail during the life of the project.
>
> The term legally does not imply that all permits needed for mining and processing have been obtained or that other legal issues have been completely resolved. However, for a reserve to exist, there should not be any significant uncertainty concerning issuance of these permits or resolution of legal issues. [Emphasis in original.]

Id. at 380. The two most significant differences between the SME and Survey definitions are: (1) SME limits the use of the terms "measured," "indicated," and "inferred" to resource determinations by eliminating the phrase "or reserve," making those terms applicable only to discussions of geologic assurance, and not applicable to economic viability; and (2) SME bases its categorization of "reserves" upon the category of the resource being considered during the course of economic viability examination. The second difference is apparent from the SME reserve classification:

> **Proven reserve.** That part of a measured resource that satisfies the conditions to be classified as a reserve.
>
> **Probable reserve.** That part of an indicated resource that satisfies the conditions to be classified as a reserve.
>
> It should be stated whether the reserve estimate is of in place material or of recoverable material. Any in-place estimate should be qualified to show the anticipated losses resulting from mining methods and beneficiation or preparation. [Bold in original.]

Id. at 380. Under the Survey's definitions a mineral resource could be classified as an "inferred reserve." SME intentionally omitted the term "reserve" from its definitions of "measured," "indicated" and "inferred," and thus the "the terms `"measured reserve,' "indicated reserve" "inferred reserve" and "possible reserve" are not part of its classification scheme. The basis for this conclusion was that the use of these terms does not convey sufficient economic assurance to be reported as a reserve. See id. at 379.

Various statutes mandate the Department's classification of mineral deposits found in public lands and impose consideration of the economic value of a mineral resource. A mining claim must contain "a valuable mineral deposit" under 30 U.S.C. § 22 (1988), and the claimant desiring to exercise a property right must show evidence of mineral of such a character that a person of ordinary prudence would be justified in the further expenditure of his labor and means, with a reasonable prospect of success, in developing a valuable mine. Castle v. Womble, 19 L.D. 455 (1894), approved in Chrisman v. Miller, 197 U.S. 313, 322 (1905). Under United States v. Coleman, 390 U.S. 599 (1968), the prudent man test

includes evidence that the mineral in question can be presently extracted, removed, and marketed at a profit.

The term "reserve" is most often employed in mining claim contests. In United States v. Feezor, supra, the Board examined how that term related to the requirement for a discovery, and the extent geologic inference could be used when determining whether a mineral resource constituted a discovery supporting the validity of a mining claim. The definitions and classification described by Survey were used.

> In United States v. Hooker, [48 IBLA 22 (1980)], the Board directly held that "indicated" reserves could be used to establish quantity and quality. Id. at 35-36; accord, United States v. Larsen, [9 IBLA 247, 261-63 (1973), aff'd, Larsen v. Morton, Civ. No. 73-19-TVC-JAW (D. Ariz. Oct. 24, 1974)]. The Board noted, however, that the question of whether "inferred" reserves could be utilized had yet to be determined. But see United States v. Wells, 11 IBLA 253, 258 (1973).
>
> As noted above, demonstrated reserves (i.e., measured and indicated) can clearly be used to show the quantity necessary to establish a discovery. We do not however, believe that any such broad ruling can be made insofar as inferred reserves are concerned. To the extent that such an estimate is based on assumed continuity or repetition for which there is geologic evidence, we feel such a reserve base can properly be considered. Where, however, a body is completely concealed, so that its actual existence must be predicated on geologic inference, use of geologic inference would, in effect, substitute for the exposure of the mineral. Such an exposure, however, is a necessary precondition to a discovery. Therefore, an "inferred" reserve whose existence is dependent solely on geologic inference cannot serve as a predicate for finding quantity and quality sufficient to support a discovery. [Emphasis in original.]

United States v. Feezor, supra at 84-85, 90 I.D. at 278. Although SME considers an "inferred reserve" lacking "the requisite degree of assurance to be reported as a reserve," Feezor recognized that there may be circumstances in which an "inferred reserve" could be considered in support of a discovery finding.

"Locatable" minerals, such as copper, gold, and silver, are generally subject to appropriation under the general mining laws if found on public lands. 30 U.S.C. § 22 (1988). When these minerals are found on acquired lands, such as those involved in this case, they are not locatable, and the right to extract these minerals must be gained pursuant to the provisions of 43 CFR Subpart 3560. See 43 CFR 3560.3-1. A prospecting permit enabling the permittee to explore for hardrock minerals may be issued when prospecting or exploration is necessary to determine the existence or workability of a particular hardrock mineral deposit. 43 CFR 3560.1(a)). Permits are issued for an initial term of 2 years, and may be extended for a period not to exceed 4 years. See 43 CFR 3562.8-1; 3562.9. A permittee who discovers a valuable deposit may apply for a preference right lease. See 43 CFR 3563.2-1, 3563.1-2(b).

When a party seeks minerals in acquired Federal lands there are several advantages gained by obtaining a prospecting permit. The rental during the initial period is lower than for leases, the permittee does not have to make a competitive bid for a lease if a valuable mineral deposit is discovered, and a prospecting permit allows mineral exploration for up to 6 years without the production and royalty requirements. In Yankee Gulch Joint Venture, 113 IBLA 106, 130-31 (1990) we held that the test used when determining whether a prospecting permit holder has discovered a valuable deposit warranting issuance of a preference right lease is almost identical to the test used when determining whether a mining claimant has discovered a valuable mineral deposit.

Although the "discovery" test may appear similar to the "workability" test, they are not the same. 6/ Under both tests there must be mineral of sufficient value to recover direct costs. Both recognize a distinction between "exploration" and "development." However, there are two significant differences. First, costs other than direct costs (such as transportation and marketing costs) are not included when making a workability determination but are included when considering whether there is a discovery. Second, a more liberal reliance on geologic inference is allowed when applying the "workability" test. 7/

In Atlas Corp., 74 I.D. 76, 79 (1967), the Department considered the relationship of these requirements in an appeal from a denial of a phosphate prospecting permit application. Atlas contended that the workability determination should be based upon "actual knowledge," i.e., geologic information gained by physical examination of the deposit within the applied for tracts by drilling or similar on the ground exploration methods, rather than through geologic inference. The decision traced the concept of workability for coal prospecting permits, and, citing Emmett K. Olson, 48 L.D. 29 (1921), concluded:

> This long continued administrative and judicial interpretation, and its recognition by Congress, is persuasive that competent evidence to establish the fact that land contains valuable deposits of certain minerals, that it is known to be valuable for minerals, that it contains commercially valuable deposits of minerals, or that exploration is not necessary to

---

6/ If the meanings were identical, one seeking to develop a deposit could obtain a prospecting permit when exploratory work would not disclose the existence or workability of a valuable deposit by presenting evidence that a known deposit was submarginal and thus would not support a "discovery."

7/ For example, a "workability" finding for some bedded minerals has been based solely on geologic inference drawn from exposures on adjacent lands. Christian F. Murer, 75 IBLA 232 (1983); Atlas Corp., 74 I.D. 76 (1967).

> determine the existence or workability of a coal or phosphate deposit, may consist of proof of the existence of the minerals in adjacent lands and of geological and other surrounding and external conditions. On the other hand, it is not necessary, as Atlas insists, to demonstrate the workability of the mineral deposit from an actual physical examination of the deposit in the land applied for by means of drilling or actual exploratory work on the ground.

Atlas Corp., supra at 84-85.

It was also noted, however, that "the character of phosphate deposits, which occur with great uniformity of thickness and consistency of quality throughout wide areas, is most similar to coal deposits." Id. at 82 n.3. Total reliance upon geologic inference may not be appropriate for other mineral deposits, such as copper, which are not typically uniform in either thickness or grade. However, if a sound basis for the inference can be demonstrated, that inference may be used. Because greater latitude in the use of geologic inference is permitted when making the workability test, it follows that evidence justifying a discovery will justify a determination that the deposit is workable. Cf. United States v. Feezor, supra (use of geologic inference for copper deposit under discovery test).

The economic component of the workability test is also different. In James C. Goodwin, 9 IBLA 139, 156-57, 80 I.D. 7, 15-16 (1973), we considered economic factors applicable to workability:

> Workability as defined by the USGS is concerned with the economics of the intrinsic factors. Extrinsic factors such as transportation, markets, etc., are not considered. However, the cost of mining must be considered. In its classification of coal lands, USGS has anticipated and assumed the ultimate coming of conditions favorable for mining and marketing of any coal if the coal is workable in terms of the intrinsic factors. In this respect, the test of workability under the Mineral Leasing Act differs from the prudent man rule under the mining laws.
>
> A further differentiation from the "prudent man" requirement of "a reasonable prospect of success" was made in Atlas Corporation, 74 I.D. 76, 84 (1967).
>
> > * * * [I]t is not necessary, in order to sustain a finding that such deposits do exist in workable quantity, that a determination can be made with some degree of assurance that a mining operation will be an economic success. Rather, it is enough that the available data is sufficient to determine that the lands under consideration would require only limited prospecting to project a program for development but

> would not require prospecting for the purpose of determining the presence of workability of the deposit. [Emphasis supplied.]

Id. at 156-57, 80 I.D. at 15-16 (emphasis supplied in Atlas Corp.). A mining claimant must show that, as a present fact, its deposit can be extracted, removed, and marketed at a profit. In Re Pacific Coast Molybdenum, Co., 75 IBLA 16, 29, 90 I.D. 352, 360 (1983); see United States v. Coleman, 390 U.S. 599, 600, 602 (1968).

Thus a showing that further information would render the workability determination more accurate is not sufficient to mandate that a decision denying a prospecting permit be vacated or reversed. The concept of workability is used to distinguish between the body of evidence which indicates that a valuable mineral deposit might be disclosed by further exploration and sufficient evidence of the existence of a mineral deposit to justify competitive bidding for the right to acquire a mineral lease. The prospecting permit is a grant of a right to explore for an undisclosed or insufficiently disclosed deposit of mineral with the exclusive right to a lease if the exploration is successful. To hold otherwise would foster the use of prospecting permits as a device for holding known but submarginal mineral deposits in anticipation of improved economic conditions. 8/

[4] By asserting that the data used by BLM was insufficient to support its finding that the deposit was workable, appellants have raised an issue of fact, but not necessarily a relevant one. Although the statement of the test for "workability" in BLM's decision might be satisfied only by a proven or probable reserve, using the SME definitions, or a measured or indicated reserve using the Survey's definitions, our analysis of decisions addressing the meaning of "workability" confirms the propriety of using inferred and subeconomic mineral resources as a basis for finding a deposit "workable," however those resources might be defined. Thus a deposit not qualifying as a "reserve" can, none the less, be classified as "workable." However, it remains proper to consider Vanderbilt's arguments to determine whether BLM's rejection of the prospecting permit application is supported by the facts.

Vanderbilt enumerates eight criticisms of BLM's report:

> 1. The BLM report considered the data from 70 exploration drill holes which were drilled on both the northern and southern portions of the property. The exploration drilling conducted to date and the information available from that drilling is simply inadequate to identify the presence of a workable deposit under the lands subject to the Permit Application.

8/ It is conceivable that the Department may prematurely deem a deposit worthy of competitive leasing. In such cases it can be expected that either no bids will be submitted or that a lease will be issued and subsequently allowed to terminate. In either event, the land will subsequently be available and the need for further exploration can be reevaluated. Present public policy favors competitive bidding over prospecting permit issuance when a deposit is known to exist.

2. Although the BLM report includes a geologic review, no geologic or statistical boundaries were placed on the grade estimations made by BLM. The failure to define the boundaries to the grade estimations allows higher grade samples to erroneously influence grade estimation in blocks beyond the reasonable geologic or statistical boundaries.

3. In determining the workability of the lands subject to the Prospecting Permit, BLM used a method of determining mineral values which included copper equivalent values. The BLM method erroneously incorporates the assumption of a correlation of the grades of molybdenum and gold to the grade of copper. The statistical analysis of the drill hole data performed by SRK clearly shows that there is no correlation between the grades of molybdenum or gold and copper. SRK Report, figures 3.1 and 3.2. The method employed by BLM to determine copper equivalent values estimates block equivalent values which have little or no relationship to the actual grades of molybdenum and gold within the blocks. The copper equivalent values can be properly estimated by determining the block grades for each element. The block grades for each mineral must then be combined to calculate a copper equivalent.

4. The inverse distance (squared) method used by BLM was inadequately applied. The BLM method did not properly incorporate a statistical analysis of the dataset as a whole with data sub-setted by rock types to determine the relationship of mineral grade distribution and geology, did not properly consider geology in the determination of samples upon which the to [sic] based estimated mineral grades for each rock type so as to prevent higher grade samples from unduly influencing large volumes of lower grade rock, did not incorporate geostatistical analysis to determine three-dimensionally the various grades for each of copper, gold and molybdenum, and did not apply three-dimensional grade distribution models to the estimation through the use of defined anisotropy.

5. The BLM's reliance upon 250 meters for the range of estimation influence is excessive. The SRK geostatistical analysis indicates that this range is excessive by 37%.

6. The inverse distance (squared) method utilized by BLM does not adequately calculate the errors inherent in the estimation. Accordingly, the inverse distance (squared) method cannot be used to determine the confidence which can be placed upon the grade estimate for each block. In contrast, Kriging techniques calculate the actual estimation errors for each block while minimizing such errors.

7. The BLM model utilized only composites from the bench for which the estimate was made and from the benches immediately

above and below the studied bench resulting in a pancake-shaped pseudo-anisotropy 500 meters in diameter and 50 meters in thickness. There is no data or analysis in the BLM report to support this.

8. The BLM report does not include a pit optimization analysis which is essential to the determination of actual costs of mining.

As noted previously, SRK was retained to review existing drill hole data and evaluate the property's mineral potential. In SRK's report, which was used as the basis for the eight objections, it summarized its findings:

> The review included geologic modeling, statistical and geostatistical analysis, economic modeling and pit optimization. The property was shown to have two distinct geochemical populations. One population had Cu grades ranging from 0.15 to 0.75% and the other had Cu grades greater than 0.75% Cu. The lack of geochemical data for block model grade estimation limited SRK's ability to develop a model which focused on the southern portion of the property where drilling was more closely spaced. Even this area of the property showed very high (30-80%) relative errors in the estimated block grade values, reflecting the quantitative deficiency of the data available.
>
> The resources totaled 180.68 million tons at average grades of 0.324% Cu, 0.0125% Mo and 0.006 oz/ton Au. Seventy percent of this resource is classified as inferred and the remaining is classified as indicated (SME, 1991 - Appendix C). Application of economic costs and values to the resource indicated the optimum pit would contain 77 million tons of ore with a NPV of -191 million dollars. These results show the existing data do not indicate the existence of a workable resource on the property.
>
> A review of a 1991 BLM study on the property indicated similar conclusions were reached by the BLM despite some flaws in the modeling techniques used. Applying the most optimistic costs used on the property to date (Duval, 1980) to our grade model, the property was still shown to be unworkable with an adjusted cash flow of -181 million dollars.
>
> Further exploration work is essential to discover heretofore unidentified resources before any prospect of a workable property is possible. Exploration work should include mapping, sampling and drilling. Until such time as further resources are discovered, it would be inappropriate to perform any further resource evaluation or prefeasibility studies.

On July 13, 1992, Vanderbilt and Teck filed another request for a hearing. By order dated July 16, 1992, we directed them to show cause why their request should not be denied by identifying the question or

uestions of fact to be specified in any order sending the case for nearing. They replied, setting forth the following issues:

1. The sufficiency of the geologic data on which BLM based its report.

2. The necessity of applying geologic or statistical boundaries for the mineral deposit and its grade.

3. The propriety of including copper equivalent values based upon the assumption of a correlation of the grades of other minerals on which the copper equivalent values are based to the grade of copper throughout the deposit.

4. The propriety and accuracy of the statistical methods used by BLM in its calculations.

5. The failure of BLM to include a pit optimization analysis in its determination of the actual cost of mining.

In its answer, BLM states that appellants' analysis "presents a unnecessarily narrow interpretation of the known information concerning the available resource," and that it is prepared to show that "limited changes in several of the variables involved will easily throw the analysis of the appellant over the line which separates a workable from an unworkable deposit" (BLM Answer at 4-5). Denny Seymour, BLM's mining engineer, reviewed the SRK report and the additional material provided by Vanderbilt. Seymour summarizes his analysis and conclusions in an affidavit attached to BLM's answer.

Seymour studied the plots of the Vanderbilt mineralized block model and concluded that Vanderbilt used less than 52 percent of the available samples with significant mineralization when estimating the available resource. Seymour also disagreed with Vanderbilt's use of a $1 per pound copper price, restating the basis for BLM's use of the 5-year average copper price of $1.10 per pound. 9/ In mining claim validity cases we have held that historic mineral values should be considered to compensate for market fluctuation. In Re Pacific Coast Molybdenum, Co., 75 IBLA at 28-29, 90 I.D. at 359-60. For example, in United States v. Crowley, 124 IBLA 374 (1992), we accepted the use of 5-year average prices for convenience. BLM admits that no profit can be shown using $1.10 per pound. However, this fact does not, per se, justify the issuance of a prospecting permit for further exploration of the deposit. The basis for BLM's holding that no further prospecting is needed is its determination that the workability of the deposit is controlled by market price, a factor which will not be altered by further prospecting.

9/ In the last 5 years the value of copper has been as high as $1.50 per pound, but its 5-year average is at $1.10 per pound.

Seymour presented four case studies based on SRK's analysis to demonstrate how close SRK's analysis is to showing the existence of a reserve 10/ (Affidavit at 13-14). The first study shows the deposit to be unworkable using SRK's assumptions, with a negative net value of over $40 million. In the second study Seymour revised capital costs for a 40,000 ton per day operation, using a 50,000 ton per day operation. Under this scenario, revenues would exceed costs by $15.1 million. In the third scenario he retained values used in the second scenario and revised the price of copper to $1.10 per pound. This increased projected net revenues to $126.3 million. Seymour's final study retained the values used in case three and updated unit mining and milling operating costs for a 40,000 ton per day operation. This resulted in reduced mining cost, increased milling cost, and net revenues of $50.8 million.

As we stated earlier, a prospecting permit may be issued when further exploration is needed to disclose a workable deposit. However, when a deposit is not workable, as appellant contends, that fact may be attributed to economic conditions rather than a insufficient knowledge regarding the anticipated size and/or grade of a mineral deposit. Vanderbilt's own analysis suggests that prospecting would not provide significant additional information which would significantly alter either the size or grade of the deposit. Thus, Vanderbilt has raised an issue of fact as to whether a reserve exists, but its submissions cannot be taken as an offer of proof that the deposit could be rendered workable by further prospecting.

BLM's answer also addressed the eight contentions Vanderbilt raised in its SOR. In response to Vanderbilt's first two contentions (the adequacy of the drill hole information BLM relied upon and BLM failure to define the geologic or statistical boundaries for its grade estimates) BLM explains that two estimates were made. Its second was based on a maximum horizontal drill hole sample value projection of 125 meters (not 250 meters). BLM also states that it found a good correlation between its geologic interpretation of the mineralized zones and Duval's. BLM states that porphyry copper deposits are relatively continuous, with a 300- to 600-foot zone of influence not being unusual, and that its use of a 125 meter zone of influence was reasonable.

Vanderbilt's third objection was to BLM's use of "copper equivalent values," which added the value of other recoverable metals. It contends that this method erroneously incorporates an assumed correlation between the contained molybdenum and gold and contained copper. BLM responds that it did not make a correlation assumption, but used the metal content reported by Duval, and that when an assay report did not list a specific metal (more likely a failure to assay for that metal than its absence) no additional value was attributed for that metal. According to BLM, this approach will usually cause the metal content to be underestimated.

10/ None of Seymour's case studies takes into account BLM's conclusion that there is substantially more recoverable mineral material than was projected by Vanderbilt or SRK's not having considered recovery of silver values.

Vanderbilt's fourth objection relates to its contention that BLM did not consider rock type distribution when making its analysis. BLM responds that during its examination it used the geologic information available to it at the time of the examination.

The fifth objection was to BLM's use of a 250 meter radius for its zone of influence. However, this radius was used for the first BLM estimate, made in 1990, not the second, which used a 125-meter radius. In response to Vanderbilt's seventh objection, that BLM's used a "pancake-shaped pseudo-anisotropy of 500 meters in diameter and 50 meters in thickness," BLM states that this criticism was directed at its 1990 resource estimate, not the 1991 estimate which employed a 125 meter search radius and a 250 meter pseudo-anisotropy diameter.

Vanderbilt's sixth objection relates to its contention that BLM did not properly apply the "inverse distance (squared) method" when making its analysis because it failed to adequately calculate the errors inherent in its estimation. It contends that Kriging techniques should have been employed to calculate the actual estimation errors for each block, minimizing these inherent errors. BLM states that, like polygon and Kriging methods, the "inverse distance squared method" is accepted in the industry. It notes that estimates using the polygon and Kriging methods involve many more hours of computer computation time, and argues that, although Kriging may be superior when developing a specific mine plan, the same degree of specificity is not required for a workability determination.

In response to Vanderbilt's eighth objection (that BLM failed to include a pit optimization analysis) BLM admits that a pit optimization analysis would be helpful, but states that it is not necessary to make this analysis when determining workability. BLM states: "Again it appears that the appellant is attempting to require the BLM to complete the type of detailed and expensive analysis that a mining company might undertake to determine exactly how it would go about developing a mineral resource" (BLM Answer at 14).

When considering what evidentiary burden should be placed upon BLM in a case such as this, it is proper to weigh that burden against the nature of the appellant's interest, and the risk that an appellant would be improperly deprived of that interest if the greater burden were not placed on BLM. 11/ BLM is not required to sustain its

---

11/ See footnote 2, supra. When procedural due process is required, a balancing of interest test is used for determining the "specific dictates" of due process: First the private interest that will be affected by the official action; second, the risk of an erroneous deprivation of such interest through the procedures used, and the probable value, if any, of additional or substitute procedural safeguards; and finally, the Government's interest, including the function involved and the fiscal and administrative burdens that the additional or substitute procedural requirement

"workability" determinations with the same quantum of evidence needed to sustain a finding that there is a "discovery." For example, physical examination of the land is unnecessary when considering an application, if the land has been classified. Christian F. Murer, 57 IBLA 333 (1981); William J. Coleman, 9 IBLA 15 (1973); J. D. Archer, 1 IBLA 26, 77 I.D. 24 (1970). When reviewing the exercise of discretionary authority to issue or deny a prospecting permit, a primary consideration is the undisputable fact that a prospecting permit applicant holds an expectancy and not an interest in the land. Thus, this balance weighs heavily against finding error because BLM failed to conduct Kriging and pit optimization analyses. There is no showing that BLM could not properly reach its conclusion without making those analyses.

Earlier in this decision, we noted that a hearing would be appropriate if there is a material issue of fact requiring introduction of testimony and other evidence. Each of the issues identified by appellant in response to our July 16, 1992, order is predicated on an incorrect understanding of the legal basis for a decision to reject a prospecting permit application. When the legal basis for the determination is clarified no material issues of fact remain. Given the discretionary nature of a decision to accept or reject a prospecting permit application, the wider scope of geologic inference allowed when determining workability, and the ability to reject a prospecting permit application for an unworkable deposit if it would be unlikely that further prospecting would establish workability, the material submitted by appellant sustains a finding that the rejection of its prospecting permit application was not arbitrary or capricious, and that the decision is supported by the record.

Therefore, pursuant to the authority delegated to the Board of Land Appeals by the Secretary of the Interior, 43 CFR 4.1, the appellant's request for a hearing is denied and the decision appealed from is affirmed as modified.

R. W. Mullen
Administrative Judge

I concur:

John H. Kelly
Administrative Judge

---

fn. 11 (continued)
would entail. Mathews v. Eldridge, 424 U.S. 319, 335 (1976). Although Vanderbilt would like to cross-examine BLM employees making the recommendation in this case, a proper application of this balancing test makes cross-examination unnecessary. BLM will satisfy its obligation if it issues a reasoned opinion which is supported by the record.

THIS PAGE
INTENTIONALLY
BLANK

Principles of a Resource/Reserve Classification for Minerals

By the U.S. BUREAU OF MINES and the U.S. GEOLOGICAL SURVEY

INTRODUCTION

Through the years, geologists, mining engineers, and others operating in the minerals field have used various terms to describe and classify mineral resources, which as defined herein include energy materials. Some of these terms have gained wide use and acceptance, although they are not always used with precisely the same meaning.

Staff members of the U.S. Bureau of Mines and the U.S. Geological Survey collect information about the quantity and quality of all mineral resources, but from different perspectives and with different purposes. In 1976, a team of staff members from both agencies developed a common classification and nomenclature, which was published as U.S. Geological Survey Bulletin 1450-A–"Principles of the Mineral Resource Classification System of the U.S. Bureau of Mines and U.S. Geological Survey." Experience with this resource classification system showed that some changes were necessary in order to make it more workable in practice and more useful in long-term planning. Therefore, representatives of the U.S. Geological Survey and the U.S. Bureau of Mines collaborated to revise Bulletin 1450-A.

Long-term public and commercial planning must be based on the probability of discovering new deposits, on developing economic extraction processes for currently unworkable deposits, and on knowing which resources are immediately available. Thus, resources must be continuously reassessed in the light of new geologic knowledge, of progress in science and technology, and of shifts in economic and political conditions. To best serve these planning needs, known resources should be classified from two standpoints: (1) purely geologic or physical/chemical characteristics–such as grade, quality, tonnage, thickness, and depth–of the material in place: and (2) profitability analyses based on costs of extracting and marketing the material in a given economy at a given time. The former constitutes important objective scientific information of the resource and a relatively unchanging foundation upon which the latter more variable economic delineation can be based.

The revised classification system, designed generally for all mineral materials, is shown graphically in figures 1 and 2 (see page 5); its components and their usage are described in the text. The classification of mineral and energy resources is necessarily arbitrary, because definitional criteria do not always coincide with natural boundaries. The system can be used to report the status of mineral and energy-fuel resources for the Nation or for specific areas.

RESOURCE/RESERVE DEFINITIONS

A dictionary definition of resource, "something in reserve or ready if needed," has been adapted for mineral and energy resources to comprise all materials, including those only surmised to exist, that have present or anticipated future value.

Resource.–A concentration of naturally occurring solid, liquid, or gaseous material in or on the Earth's crust in such form and amount that economic extraction of a commodity from the concentration is currently or potentially feasible.

Original Resource.–The amount of a resource before production.

Identified Resources.–Resources whose location, grade, quality, and quantity are known or estimated from specific geologic evidence. *Identified resources* include economic, marginally economic, and subeconomic components. To reflect varying degrees of geologic

(**Identified Resources**–Continued)

certainty, these economic divisions can be subdivided into *measured, indicated,* and *inferred.*[1]

Demonstrated.–A term for the sum of *measured* plus *indicated.*

Measured.–Quantity is computed from dimensions revealed in outcrops, trenches, workings, or drill holes; grade and(or) quality are computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are spaced so closely and the geologic character is so well defined that size, shape, depth, and mineral content of the resource are well established.

Indicated.–Quantity and grade and(or) quality are computed from information similar to that used for measured resources, but the sites for inspection, sampling, and measurement are farther apart or are otherwise less adequately spaced. The degree of assurance, although lower than that for measured resources, is high enough to assume continuity between points of observation.

Inferred.–Estimates are based on an assumed continuity beyond measured and(or) indicated resources, for which there is geologic evidence. *Inferred resources* may or may not be supported by samples or measurements.

Reserve Base.–That part of an identified resource that meets specified minimum physical and chemical criteria related to current mining and production practices, including those for grade, quality, thickness, and depth. The *reserve base* is the in-place demonstrated (measured plus indicated) resource from which reserves are estimated. It may encompass those parts of the resources that have a reasonable potential for becoming economically available within planning horizons beyond those that assume proven technology and current economics. The *reserve base* includes those

[1]The terms "proved," "probable," and "possible", which are commonly used by industry in economic evaluations of ore or mineral fuels in specific deposits or districts, have been loosely interchanged with the terms *measured, indicated,* and *inferred.* The former terms are not a part of this classification system.

(**Reserve Base**–Continued)

resources that are currently economic (*reserves*), marginally economic (*marginal reserves*), and some of those that are currently subeconomic (*subeconomic resources*). The term "geologic reserve" has been applied by others generally to the *reserve-base* category, but it also may include the *inferred-reserve-base* category; it is not a part of this classification system.

Inferred Reserve Base.–The in-place part of an identified resource from which inferred reserves are estimated. Quantitative estimates are based largely on knowledge of the geologic character of a deposit and for which there may be no samples or measurements. The estimates are based on an assumed continuity beyond the reserve base, for which there is geologic evidence.

Reserves.–That part of the reserve base which could be economically extracted or produced at the time of determination. The term *reserves* need not signify that extraction facilities are in place and operative. *Reserves* include only recoverable materials; thus, terms such as "extractable reserves" and "recoverable reserves" are redundant and are not a part of this classification system.

Marginal Reserves.–That part of the reserve base which, at the time of determination, borders on being economically producible. Its essential characteristic is economic uncertainty. Included are resources that would be producible, given postulated changes in economic or technologic factors.

Economic.–This term implies that profitable extraction or production under defined investment assumptions has been established, analytically demonstrated, or assumed with reasonable certainty.

Subeconomic Resources.–The part of identified resources that does not meet the economic criteria of reserves and marginal reserves.

Undiscovered Resources.–Resources, the existence of which are only postulated, comprising deposits that are separate from identified resources. *Undiscovered resources* may be postulated in deposits of such grade and physical location as to render them economic, marginally economic, or subeconomic. To reflect varying degrees of geologic certainty,

(**Undiscovered Resources**—Continued)

undiscovered resources may be divided into two parts:

Hypothetical Resources.—Undiscovered resources that are similar to known mineral bodies and that may be reasonably expected to exist in the same producing district or region under analogous geologic conditions. If exploration confirms their existence and reveals enough information about their quality, grade, and quantity, they will be reclassified as identified resources.

Speculative Resources.—Undiscovered resources that may occur either in known types of deposits in favorable geologic settings where mineral discoveries have not been made, or in types of deposits as yet unrecognized for their economic potential. If exploration confirms their existence and reveals enough information about their quantity, grade, and quality, they will be reclassified as identified resources.

Restricted Resources/Reserves.—That part of any resource/reserve category that is restricted from extraction by laws or regulations. For example, *restricted reserves* meet all the requirements of reserves except that they are restricted from extraction by laws or regulations.

GUIDELINES FOR CLASSIFICATION OF MINERAL RESOURCES

1. All naturally occurring metals, nonmetals, and fossil fuels in sufficient concentration can be classified in one or more of the categories.

2. Where the term *reserves* is used alone, without a modifying adjective such as indicated, marginal, or inferred, it is to be considered synonymous with the demonstrated-economic category, as shown in figure 1.

3. Definitions of resource categories can be modified for a particular commodity in order to conform with accepted usage involving special geological and engineering characteristics. Such modified definitions for particular commodities will be given in forthcoming government publications.

4. Quantities, qualities, and grades may be expressed in different terms and units to suit different purposes, but usage must be clearly stated and defined.

5. The geographic area to which any resource/reserve estimate refers must be defined.

6. All estimates must show a date and author.

7. The *reserve base* is an encompassing resource category delineated by physical and chemical criteria. A major purpose for its recognition and appraisal is to aid in long-range public and commercial planning. For most mineral commodities, different grades and tonnages, or other appropriate resource parameters, can be specified for any given deposit or area, or for the Nation, depending on the specific objectives of the estimators; therefore, the position of the lower boundary of the reserve base, which extends into the subeconomic category, is variable, depending on those objectives. The intention is to define a quantity of in-place material, any part of which may become economic, depending on the extraction plans and economic assumptions finally used. When those criteria are determined, the initial reserve-base estimate will be divided into three component parts: reserves, marginal reserves, and a remnant of subeconomic resources. For the purpose of Federal commodity assessment, criteria for the reserve base will be established for each commodity.

8. *Undiscovered resources* may be divided in accordance with the definitions of *hypothetical* and *speculative resources,* or they may be divided in terms of relative probability of occurrence.

9. *Inferred reserves* and the *inferred reserve base* are postulated extensions of reserves and of the reserve base. They are identified resources quantified with a relatively low degree of certainty. Postulated quantities of resources not based on reserve/reserve-base extensions, but rather on geologic inference alone, should be classified as undiscovered.

10. Locally, limited quantities of materials may be produced, even though economic analysis has indicated that the deposit would be too thin, too low grade, or too deep to be classified as a reserve. This situation might arise when the production facilities are already established or when favorable local circumstances make it possible to produce material that elsewhere could not be extracted profitably. Where such production is taking place, the quantity of in-place material shall be included in the reserve base, and the quantity that is potentially producible shall be included as a reserve. The profitable production of such materials locally, however, should not be used as a rationale in other

areas for classifying as reserves, those materials that are similar in thickness, quality, and depth.

11. Resources classified as reserves must be considered economically producible at the time of classification. Conversely, material not currently producible at a profit cannot be classified as reserves. There are situations, however, in which mining plans are being made, lands are being acquired, or mines and plants are being constructed to produce materials that do not meet economic criteria for reserve classification under current costs and prices, but would do so under reasonable future expectations. For some other materials, economic producibility is uncertain only for lack of detailed engineering assessment. The marginal-reserves category applies to both situations. When economic production appears certain for all or some of a marginal reserve, it will be reclassified as reserves.

12. Materials that are too low grade or for other reasons are not considered potentially economic, in the same sense as the defined resource, may be recognized and their magnitude estimated, but they are not classified as resources. A separate category, labeled *other occurrences,* is included in figures 1 and 2.

13. In figure 1, the boundary between *subeconomic* and *other occurrences* is limited by the concept of *current or potential feasibility of economic production,* which is required by the definition of a resource. The boundary is obviously uncertain, but limits may be specified in terms of grade, quality, thickness, depth, percent extractable, or other economic-feasibility variables.

14. Varieties of mineral or energy commodities, such as bituminous coal as distinct from lignite, may be separately quantified when they have different characteristics or uses.

15. The amount of past cumulative production is not, by definition, a part of the resource. Nevertheless, a knowledge of what has been produced is important to an understanding of current resources, in terms of both the amount of past production and the amount of residual or remaining in-place resource. A separate space for cumulative production is shown in figure 1. Residual material left in the ground during current or future extraction should be recorded in the resource category appropriate to its economic-recovery potential.

16. In classifying reserves and resources, it is necessary to recognize that some minerals derive their economic viability from their coproduct or byproduct relationships with other minerals. Such relationships must be clearly explained in footnotes or in an accompanying text.

17. Considerations other than economic and geologic, including legal, regulatory, environmental, and political, may restrict or prohibit the use of all or part of a deposit. Reserve and resource quantities known to be restricted should be recorded in the appropriate classification category; the quantity restricted and the reason for the restriction should be noted.

18. The classification system includes more divisions than will commonly be reported or for which data are available. Where appropriate, divisions may be aggregated or omitted.

19. The data upon which resource estimates are based and the methods by which they are derived are to be documented and preserved.

RESOURCES OF (commodity name)

[A part of reserves or any resource category may be restricted from extraction by laws or regulations (see text)]

AREA: (mine, district, field, State, etc.) UNITS: (tons, barrels, ounces, etc.)

Cumulative Production	IDENTIFIED RESOURCES			UNDISCOVERED RESOURCES	
	Demonstrated		Inferred	Probability Range (or)	
	Measured	Indicated		Hypothetical	Speculative
ECONOMIC	Reserves		Inferred Reserves		
MARGINALLY ECONOMIC	Marginal Reserves		Inferred Marginal Reserves		
SUB-ECONOMIC	Demonstrated Subeconomic Resources		Inferred Subeconomic Resources		

Other Occurrences	Includes nonconventional and low-grade materials

Author: Date:

FIGURE 1.—Major elements of mineral-resource classification, excluding *reserve base* and *inferred reserve base*.

RESOURCES OF (commodity name)

[A part of reserves or any resource category may be restricted from extraction by laws or regulations (see text)]

AREA: (mine, district, field, State, etc.) UNITS: (tons, barrels, ounces, etc.)

Cumulative Production	IDENTIFIED RESOURCES			UNDISCOVERED RESOURCES	
	Demonstrated		Inferred	Probability Range (or)	
	Measured	Indicated		Hypothetical	Speculative
ECONOMIC	Reserve		Inferred		
MARGINALLY ECONOMIC	Base		Reserve Base		
SUB-ECONOMIC					

Other Occurrences	Includes nonconventional and low-grade materials

Author: Date:

FIGURE 2.—*Reserve base* and *inferred reserve base* classification categories.

☆ U.S. GOVERNMENT PRINTING OFFICE: 1980 O— 311-344/106

THIS PAGE
INTENTIONALLY
BLANK

WHY GEOSTATISTICS?

A. G. Royle, senior lecturer, Department of Mining and Mineral Sciences, University of Leeds, England

Evaluators of mines have always had an active appreciation of the effects of "regionalized variables"—those variables whose values are related in some way to their positions. In mineral deposits, the occurrence of high-grade and low-grade sections, pay shoots, and lower-grade fringe areas make ore grade a regionalized variable.

Mining engineers and mine geologists have long realized the uselessness of taking a large number of samples at one ore exposure, which may occur in a persistently poor or rich zone and tell little of the nature of ore elsewhere in the deposit. Furthermore, whenever circumstances have allowed, mine technicians have taken samples at regular intervals, rather than deliberately "randomizing" their data by taking samples at randomly selected sites.

Such sampling practice was well-founded, but a theoretical demonstration of its soundness was difficult. Also, when things went wrong, corrections were commonly made by applying "experience" factors. Using such methods, payable deposits might be written off, while other deposits might be overvalued and never deliver estimated grades to the mill. "Overvaluation of a deposit by diamond drilling," is a well-known phrase. But the diamond drill is neutral, so the cause must lie elsewhere.

REGIONALIZED VARIABLES

Assume that the following sets of numbers represent the values of two series of samples taken along drives in different orebodies:

1 7 3 6 2 9 4 8 5

1 3 5 7 9 8 6 4 2

The samples have the same mean value, 5.0, and the same variance, 6.67, but they obviously come from two distinct kinds of mineralization.

Variance is calculated by the equation

$$V = \frac{(MV - SV(1))^2 + (MV - SV(2))^2 \ldots + (MV - SV(n))^2}{\text{No. of samples } (n)}$$

where V = variance, MV = mean value, and SV = sample value.

The first of the two series cited as examples would probably be called very erratic and the second more uniform. In other words, the single statistic of variance is of little value by itself. If a geologist were told that the mean and variance of the contained metal percentage in the samples were 5.0 and 6.7 respectively, he would have no idea whether the mineralization was erratic or otherwise.

Next, suppose that the following set of numbers represents a line of samples in another part of the uniform deposit:

31 33 35 37 39 38 36 34 32

They have a mean of 35, but their variance is 6.67—the same as the first two sets of samples. More important, they have the same *spatial variation* as the second set, as measured by the differences between the sample values at various sampling intervals. The following calculations emphasize this point.

At one sampling interval, the average squared difference between sample values is

$$\frac{(31-33)^2 + (33-35)^2 + \ldots (36-34)^2 + (34-32)^2}{8}$$

$$= 3.63$$

At two sampling intervals, the average squared difference becomes

$$\frac{(31-35)^2 + (33-37)^2 + \ldots (38-34)^2 + (36-32)^2}{7}$$

$$= 12.86$$

Fig. 1—A typical variogram

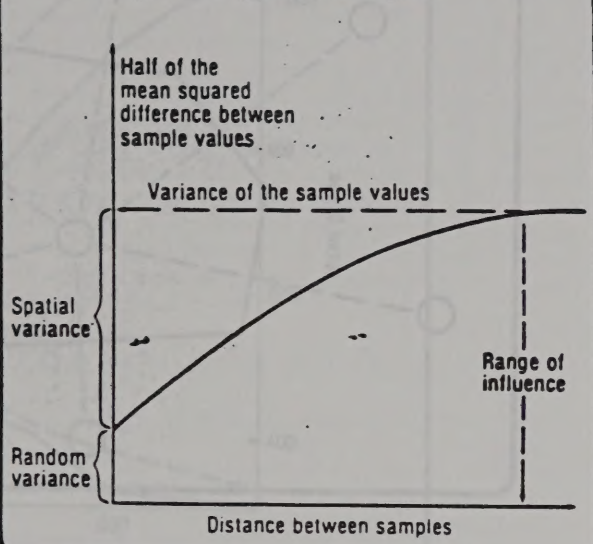


Fig. 2—Range of influence in a gold mine

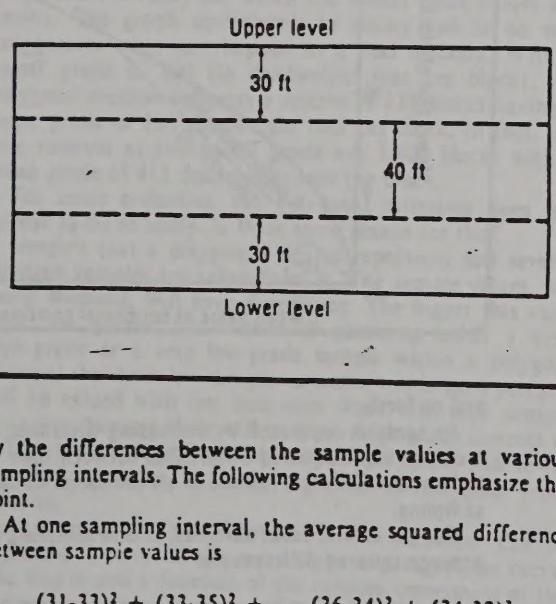


Fig. 3—Polygon construction for the method of polygons

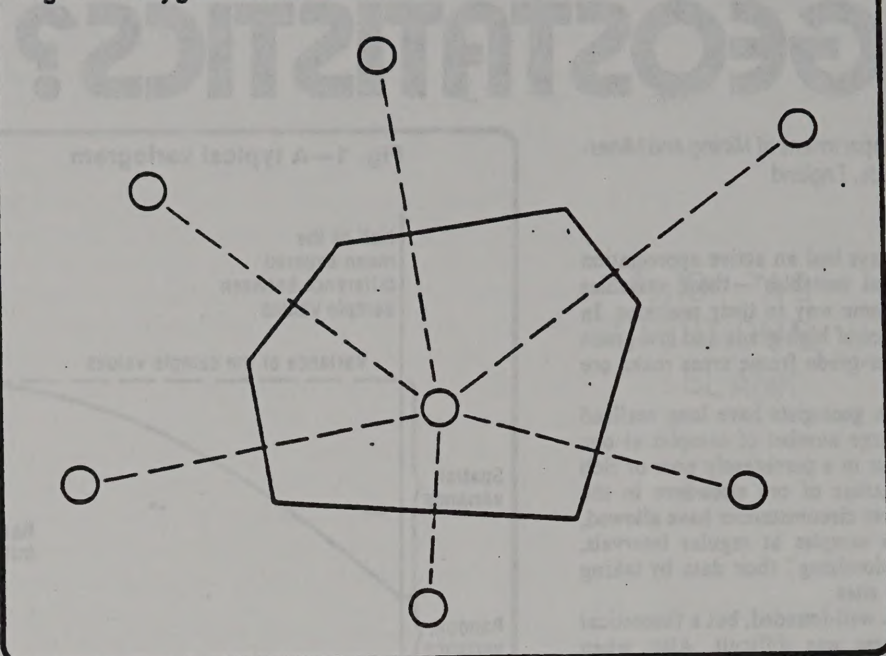
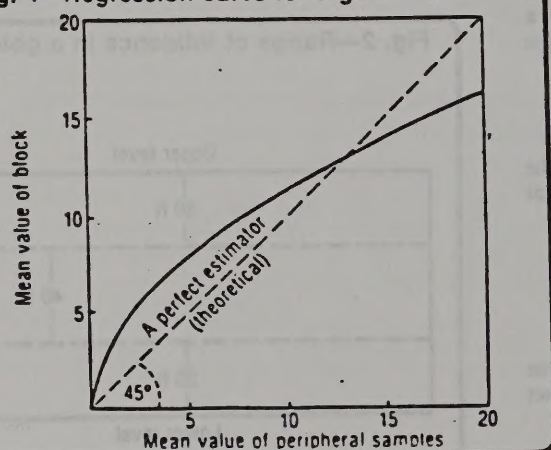


Fig. 4—Regression curve for a gold mine



and so forth.

As samples separated by increasing distances are considered, the mean squared differences between their values tend to increase. This is a feature of all orebodies at some scale of sampling.

For the erratic set of samples, at one sampling interval the average squared difference is

$$\frac{(1-7)^2 + (7-3)^2 \ldots + (4-8)^2 + (8-5)^2}{8} = 22.0$$

This value is larger than the value of the more uniform samples above—in other words, much more erratic.

As will be demonstrated in later articles in this series, these mean squared differences can be used to construct a variogram depicting the spatial variability at increasing distances between sample points. First, there is a range corresponding to the "range of influence" of the variable, a well-known concept that the variogram puts on a rational and numerical basis (Fig. 1). Second, the variogram splits the total variance into two parts. One represents the "spatial" differences between the values of samples taken at points separated by increasingly larger distances. The other represents local or short-range variances.

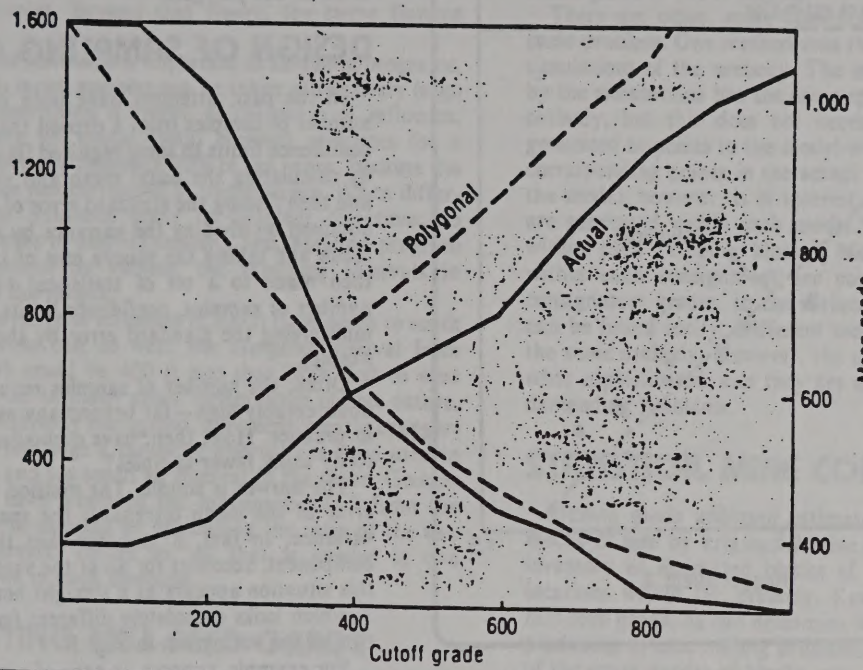
The latter can best be appreciated by considering the assay values for the two halves of a split drill core. For some deposits, the differences between such assay values are small—usually the case with base metals. In gold deposits, they can be large. This local, random variance is called the nugget variance, while the larger-scale one is the spatial variance. Thus, thicknesses, assays, foot-percent values, and other values measured in mineral deposits exhibit a partly random variation and a partly spatial variation. Fig. 1 is a typical variogram that illustrates all of these points.

VALUATION OF ORE BLOCKS

Underground blocks of ore are often valued by taking samples on the levels running along the upper and lower edges of the blocks and using the mean of the samples, probably weighted by their thicknesses, as an estimate of the mean value of the stope. Consider the results of such an estimating procedure for a gold-quartz deposit in which the range of influence of sample values is 30 ft and the level interval is 100 ft (Fig. 2). Obviously nothing will be known about the central, 40-ft-thick slice through the stoping block, and taking more samples on the upper and lower levels will not improve the estimates because the slice is out of range of all the samples. A sublevel through the center of the stope would be a better approach.

For base metals deposits, where the range of influence may be several hundred feet, the samples on the levels might be adequate. Indeed, some mines could take fewer samples without any serious loss of accuracy in estimation. Using geostatistics, the likely errors of estimation in such valuations can be calculated very quickly.

Fig. 5—Bias in the method of polygons



THE METHOD OF POLYGONS

The method of polygons is widely used to estimate total reserve in a deposit. Basically, the method divides the deposit into a series of polygons centered on individual samples, the polygons being constructed by drawing in the perpendicular bisectors of the lines between sampling points (Fig. 3). Each polygon is assumed to have a mean value equal to the value of the sample it contains. Until fairly recently, very few studies had been devoted to the errors inherent in this method.

In view of the extent to which the polygon method has been used, it may be worthwhile to review in very general terms the "regression effect"—the undervaluation that results when a low-value set of samples is used to value a block of ground and the overvaluation if a high-value set is used. An actual regression curve from a gold mine is shown in Fig. 4. The mean value of samples taken around the periphery of a stoping block at the mine is the estimator for the block. For example, a sample set with a mean value of 2.0 will, on average, be obtained for blocks having a mean value of 5.1, while an above-average sample set of 17.0 indicates blocks having a value of only 14.6.

If a perfect estimator were available, the regression curve would be a straight line sloping at 45° to each axis, and all points would lie on the line. As will be demonstrated later, the bias of an estimator can be measured in terms of the mean slope of the regression curve.

How does the bias of the polygonal estimator arise, and can this bias explain overvaluation resulting from use of the estimator? First of all, if good core recovery is achieved and if assays are correct, then the values obtained for the drill cores are valid. However, while they are true values for the cores, they are not true for the polygons. A low-value core sample will lie below the true mean of its polygon, and a high-value sample will similarly lie above it—which is the bias of the method. If such values are taken to represent reserves, estimated tonnages will be either high or low, according to the bias.

An example of bias is illustrated in Fig. 5, which is based on a model orebody for which the actual block values are known. The graph corresponds to reality and in no way exaggerates what can happen in a real orebody. With a cutoff grade of 200 (in deadweight tons per block), the polygonal method indicates a reserve of 935 blocks having a mean grade of 530 deadweight tons per block. In fact, the true reserves at this cutoff grade are 1,400 blocks with a mean grade of 415 deadweight tons per block.

For some orebodies, the polygonal estimator does not appear to do so badly. Is there some reason for this?

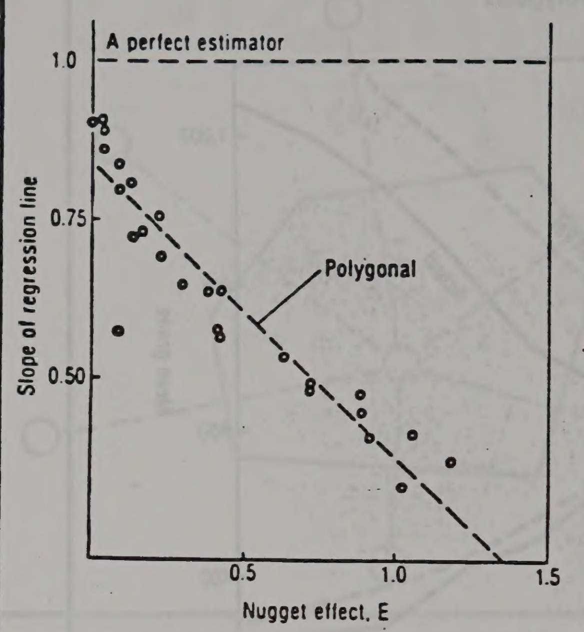
Imagine that a polygon is drilled repeatedly and several hundred samples are taken from it. The sample values, not being identical, will have a variance. The bigger this variance, the greater the chance of obtaining either a very high-grade or a very low-grade sample within a polygon. Deposits that have low sample variances within each polygon will be valued with less bias than those with high sample variances, because there will be fewer high-grade samples to inflate the expected mean grades of polygons, and fewer low-grade samples to cause rejection of polygons from reserves.

As mentioned above, the bias of an estimator can be expressed in terms of the mean slope of a regression curve. The bias is also a function of the random component of the variogram and can be expressed in terms of the nugget effect E:

$$E = \frac{\text{Random component of the variance}}{\text{Spatial component of the variance}}$$

The slope of the regression line is plotted against the nugget effect in Fig. 6. Remembering that a perfect estimator will have a regression slope of 1 (= tan 45°), one can see that some bias is still present even for very low values of E. Also, results derived by the polygonal method worsen rapidly as E increases. Thus, the method is virtually useless for

Fig. 6—Bias using the nugget effect



"erratic" deposits, or—in geostatistical terms—for deposits having high nugget effects.

The range of influence also has an effect on results derived by the polygonal method. If the range of influence is small, leaving the outer areas of the polygons beyond the range of influence of their central samples, then the values of the samples will be unrelated to the outer areas. (See Fig. 2.) Even if several samples are taken inside each polygon and averaged, there is still some residual bias, although it is reduced.

As will be shown later in this series, the practice of "kriging" overcomes the regression effect by valuing a polygon through the use of both its own value and the sample values of the surrounding polygons. Sample values are weighted to minimize errors of estimation and produce an unbiased estimate. The weights have nothing to do with the areas of the polygons.

DESIGN OF SAMPLING PROGRAMS

In the past, attempts have been made to determine the number of samples from a deposit that are needed to reduce confidence limits to some required figure. This has been done by calculating the usual mean and variance of the samples and then finding the standard error of the mean. The latter is obtained by dividing the variance by the number of samples taken and taking the square root of the result. Reference is then made to a set of statistical *t*-tables. For any given number of samples, confidence limits can be determined by multiplying the standard error by the number found in the *t*-tables.

Often, the number of samples required by this method is unacceptably high—far beyond any number likely to be used in practice. How, then, have orebodies been valued satisfactorily using fewer samples?

The answer is simple. The method that uses the standard error of the mean disregards the spatial component of the variance. In fact, it assumes that the nugget, or random component, accounts for all of the variance. On a variogram, this situation appears as a straight horizontal line, as in Fig. 7, which looks completely different from the probable structure of the variogram in Fig. 1.

For example, suppose an area of ground 2,000 ft x 2,000 ft square is to be valued by samples taken regularly within it. If only four samples were to be taken, each sample would be at the center of a 500 x 500-ft square, and so on. Assume a random (nugget) component of variance of 30 units, a spatial component of 150 units, and a range of influence of 500 ft. Fig. 8 shows the number of samples required to produce a given geostatistical standard deviation or a given standard error of the mean for nonspatial statistics. One can see how an appreciation of the effect of the spatial component of the total variance reduces the number of samples needed in practice for any desired confidence limits. Calculations of this kind require only a few minutes with a pocket calculator and a set of geostatistical tables.

Fig. 8 also shows that a point is soon reached at which

Fig. 7—Effect of overemphasis of nugget variance

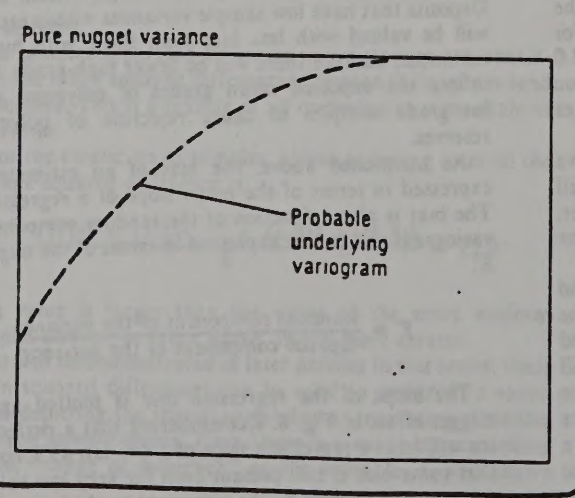
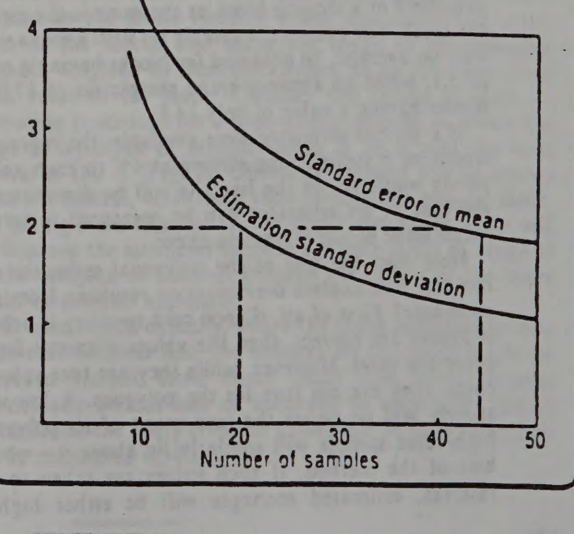


Fig. 8—How spatial component reduces sampling requirements



taking of further samples has only a very small effect on reducing the confidence limits. In this example, it occurs at about 30 samples. Beyond that figure, the curve flattens markedly.

The range of influence is important in sampling programs, and when it is large, samples can be taken at relatively large intervals. If something is known of the range of influence, sample spacing for global (total reserve) estimates for a deposit may initially approach 90% of the range, because the samples will be just correlated at this distance. When different ranges of influence occur in different directions, as commonly occurs in alluvial deposits, the sampling program can be optimized by varying the sampling intervals in proportion to the ranges.

Thus, if the range of influence is 500 ft from north to south and 250 ft from east to west, the sampling interval from north to south could be 400 ft and that from east to west could be 200 ft. In many cases, anisotropies of this nature, although qualitatively recognized, have been greatly exaggerated because too large a sampling interval has been used in one direction and too small an interval in the other. Geostatistics would have produced the maximum effect from the same number of samples by optimizing the ratio of the two sampling intervals. To do so, all that is needed is two variograms, one in each direction.

FLUCTUATING MILL-FEED GRADE

It is a mistake to believe that fluctuations in mill-feed grade arise only from errors of estimation of block mean grades or that they arise solely from differences between the mean values of consignments of ore sent to the mill. In fact, fluctuations arise from both sources, and geostatistics allows them to be calculated fairly easily. Fluctuations in the true mean grades of consignments of ore sent to the mill depend on two things, apart from the natural variation of grade within the deposit. In any given deposit, such fluctuations depend also on the size of the blocks for which the mean grades have been estimated and on the size of the consignments of ore removed from the blocks and delivered to the mill.

As usual, these fluctuations can be expressed as a variance. The mean grade of the consignments will be the mean grade of the block, but the grades of the individual consignments will have a scatter, or variance, about the mean value. If the consignments are equivalent to a small panel measuring ($l \times h$) and the stoping block, whose mean grade has been estimated by kriging, measures ($L \times H$), then the variance σ_b^2 of the consignment grades within the block is simply:

$$\sigma_b^2 = F(H, L) - F(h, l)$$

and the F functions in the equation can be found in a set of geostatistical tables or from graphs.

Thus, the total variance of fluctuation is:

$$\sigma_f^2 = \sigma_b^2 + \sigma_k^2 \text{ (kriging variance)}$$

so it is possible to determine how many blocks must be mined simultaneously to keep the grade fluctuations of the mill feed between any given limits for any set percentage of feed time. Assuming a normal distribution for σ_f^2, the mill feed fluctuation will be

$$Z^* \pm t\sqrt{\frac{\sigma_f^2}{N}}$$

where t is a multiplier found in the t-tables for the appropriate time percentage (such as 90% or 95%), N is the number of blocks mined together, and Z^* is the estimated mean grade of the N blocks.

There are other, more complicated ways of tackling the same problem. One method uses the variogram as a basis for simulations of the orebody. The model orebody represented by the simulations has the same spatial variation as the real orebody, but this does not necessarily imply that grades generated at points in the model will be the same as those at corresponding points in the actual orebody. Variance within the model, however, is of interest. Large numbers of points are generated within each model "consignment" and hence within each "stoping block." Mean grades of the points within each consignment are considered to be the mean consignment grades, so the variance of these within a block can be found easily. Different models can be generated for the same orebody. However, the use of simulations is somewhat sophisticated, and they are often applied only to more demanding problems.

STATISTICAL MINE CONTROL

Having made unbiased estimates of the mean values of blocks of ore by kriging, a mine planner has available an inventory of estimated blocks of ore, and he knows their locations within the orebody. Knowing the fluctuations in mill-feed grade, he can determine how many stopes should be producing to ease milling problems. With accurate estimates of the mean grades of blocks, rational production scheduling can begin.

Above all, such scheduling is based on unbiased estimates. By using the kriging variances of estimation, a planner can calculate the probability of failing to achieve a certain production over a given mining period. (The great majority of production literature rarely embarks on this consideration.)

Geostatistics can also prevent conflicts among various operating departments. For example, if production is lagging, the miner typically blames the geologist for "finding" nonexistent ore or for overestimating grade, while the mill blames the miner for delivering low-grade ore or for failing to keep up production. The geologist, on the other hand, accuses the miner of missing ore or of overbreaking waste, and the miner accuses the mill of hiding its deficiencies in what appear to be unnecessarily complicated calculations.

All of these disagreements can be avoided by use of geostatistics. Variations in ore grade can be explained, estimation variances will predict the quality of estimates, and straightforward calculations can be made of the probability of delivering—or not delivering—expected grades or tonnages. If in fact predicted grades or tonnages are persistently wrong, geostatistics will more easily unmask the sources of errors.

For example, an estimation method that does not use geostatistics might on a given occasion undervalue ore grade and overvalue tonnage. If under these conditions a miner appeared to be achieving the required grades and tonnages, dilution by overbreaking of waste obviously would be occurring. But because the errors would compensate for each other, they would go unnoticed. The unbiased estimates of geostatistics would sort out such a situation.

WHY GEOSTATISTICS?

A brief argument for the use of geostatistics would make the following points:

- Geostatistics is soundly based on good valuation practice, in that it provides a rational theoretical basis for intuitive mine valuation practices of the past.
- Recognition of the fact that total variation is partly

random and partly spatial leads to estimates that are unbiased and have lower errors of estimation. Geostatistics also shows where to look for sources of errors.

- Geostatistics explains why some traditional types of estimator, such as the polygonal method, produce biased estimates. These biases may be removed by using other kinds of estimators, or kriging.
- Grade-control problems are amenable to geostatistical treatment because the number of working stopes needed to keep mill-feed grade fluctuations within predetermined limits can be easily determined.
- Valuation and mining operations can be kept under statistical control, and deviations, which must occur with some calculable frequency, can be forecast and dealt with. The unbiased nature of geostatistical estimations is of prime importance for such control. They do not create compensating errors that mask other deficiencies in ore estimation, mining, or milling.
- Sampling and valuation programs can be designed economically, and geostatistics can help optimize sampling patterns.

As for the mathematical difficulties of geostatistics, they certainly exist at the theoretical level and in the more advanced practical applications, but a great deal of good valuation practice is within the capability of a person with only a low level of mathematical expertise. Geostatistical tables are available that list possibly dismaying multiple integrals as simple arithmetic numbers. The majority of orebodies can be valued effectively by a competent geologist or engineer who has a working knowledge of geostatistics.

One other criticism is sometimes leveled: "Geostatistical methods need more data to make them work." This view is a sure sign that someone has a firm grasp on the wrong end of the stick. When geostatistics appears not to work, the reason is usually precisely that the data were insufficient in the first place. Polygons, for example, can be drawn about any pattern of boreholes, but as usually practiced the polygonal method does not produce an estimate of the errors of estimation nor show whether there is any correlation between sample values at the sampling interval used. In other words, it does not show whether a "valuation" in an acceptable sense has been made.

By contrast, the first step in using geostatistics would be to produce a variogram of sample values or thicknesses. If this appeared to be random (the pure nugget variance of Fig. 7), one could conclude that the sampling interval was too large for the purpose of assigning values to blocks of ground between the boreholes—or for designing a mining program to extract blocks above some given cutoff grade. At some mines, "high grading" of blocks erroneously thought to contain exceptionally high values has proved not only disappointing but disastrous.■

ABOUT THIS SERIES

Miners and geologists have always known that the value of minerals in a given volume of ore depends heavily on the position of the ore in the orebody and on the value of the ground surrounding it. Thus, traditional methods of ore reserve estimation have attempted to combine data on the position of the sample with an intuitive notion of "area of influence" to produce usable results. Polygonal and triangular weighting, rectangular zones of influence, and inverse distance methods were all developed so that *both* characteristics—spatial position and value of surrounding ground—would be included in the estimation. However, there is no objective way to measure the reliability of these estimating techniques.

In an attempt to produce estimators for which confidence limits might be derived, workers in the field have borrowed or developed techniques based on formal statistical theory. Unfortunately, these techniques were originally developed for random observations of independent individuals in a given population, with no regard for spatial position.

Therefore, both approaches have to some extent fallen short. The classical approach embodies the notion of the spatial variation of values at the expense of quantitative measures of reliability. The formal statistical methods produce confidence limits but ignore spatial relationships.

Following earlier empirical work by Krige, Matheron developed the theory of regionalized variables specifically to resolve these problems.

A random variable is a measurement of individuals that is expected to vary between individuals in some probabilistic way. A regionalized variable is a random variable that takes different values according to its *position* within some region. In ore reserve estimation, the orebody is the region, and grade, thickness of ore, density, or any other desired measure are the regionalized variables. The behavior of ore grades within any given deposit conditions this method of reserve estimation to that particular deposit. The continuity and/or erratic nature of the mineralization, its variation in grade over the deposit, and the positions of the available samples combine to create a unique estimating technique for each deposit. The technique produces not only the best possible estimates of ore reserves, both locally and throughout the deposit, but also yields a direct quantitative measure of the reliability of the estimates.

This series of articles will describe more fully the need for a geostatistical approach to ore analysis and the way in which such an analysis is performed. The reader is assumed to be familiar with some elementary statistical terms, such as mean, variance, and probability distribution, but the specialized vocabulary has been kept to a minimum.

Later articles in the series will be written by Isobel Clark, lecturer, Department of Mineral Resources Engineering, Royal School of Mines, London; Peter Brooker, senior lecturer, Department of Economic Geology, University of Adelaide, Australia; Jean-Michel Rendu, associate professor, Department of Metallurgical and Mineral Engineering, University of Wisconsin-Madison; Harry Parker, chief geologist and statistician, Fluor Mining and Metals Inc.; Robert Sandefur, development engineer, Utah International Inc.; and Andre Journel, visiting professor, Department of Applied Earth Sciences, Stanford University.

The preparation of the series was coordinated by Pierre F. Mousset-Jones, associate professor, Department of Mining Engineering, Mackay School of Mines, University of Nevada, Reno.

THE SEMIVARIOGRAM — PART 1

Dr. Isobel Clark, lecturer, Department of Mineral Resources Engineering, Royal School of Mines, London

The procedure for making a geostatistical ore reserve estimation can be divided into two parts. The first is the investigation and modeling of the physical and statistical structure of the orebody for which the estimation is being made. Concepts of continuity and structure in the deposit are embodied in "semivariograms" that are constructed during this first step. The second stage of the procedure is the estimation process itself—kriging—which depends entirely on the semivariograms constructed during the first stage.

This article and the next in this series will cover the first stage of the geostatistical estimating procedure. In this article, the theoretical foundations of geostatistics are discussed without resort to mathematical equations. (The reader is assumed to be familiar with such terms as probability distribution, mean, and variance.) Other statistical approaches to ore reserve estimation are reviewed, and the development of geostatistics from these is shown as the next logical step. The importance of the semivariogram is a direct product of these basic assumptions. Practical methods for constructing a semivariogram, in both ideal and slightly more realistic situations will be described, as will the intuitive interpretation of the form of the graph. Finally, the most common types of simple semivariograms will be described, along with the "models" that correspond to them.

CONCEPTS OF GEOSTATISTICS

Like all statistical techniques, geostatistics is based on a probabilistic concept. Such concepts and assumptions have been described often in mathematical terms[1] but hardly ever in terms of their practical implications.

Suppose a set of samples is taken from points scattered over a deposit. Classical statistical techniques, for example Sichel's *t* estimator, assume that all of the samples are taken randomly and independently from one simple probability distribution—that all points in the deposit can be thought of as samples from that single distribution. This assumption is called stationarity. The approach does not involve any knowledge of the actual sample positions or of relationships between samples, but it can be extremely valuable as a method for gaining an overall picture of the deposit, or part of the deposit, especially at the prefeasibility stage.

Trend analyses—rolling means, trend surfaces, etc.—allow for spatial structure by altering slightly the above assumption. These methods assume that each point is a sample of the same type of distribution but that the average of the distribution can change from place to place in the deposit. This trend is a change in expected grade, and actual grade is a random sample from a distribution having the expected grade as its mean. Thus, a trend surface will never go through data values, because there will always be some random variation about the mean. Polynomial trend surfaces try to approximate these expected values by a low order polynomial equation; rolling means try to approximate them by an averaging process over a local area.

The assumptions of geostatistics are not so restrictive nor so easy to explain. Geostatistics accepts the concept that each point in the deposit represents a sample from some distribution, but the distribution at any one point may differ completely from that at all other points in the deposit in its form, mean, and variance.

Consider two points separated by a distance *h* and having a given relative orientation. If the difference in grades is taken between these two points, then this difference will be a variable that follows a distribution dictated by the distributions at each of the two points. Take another pair of points the same distance apart and having the same orientation. The difference in grade between these points will also have a distribution.

Geostatistics assumes that the distribution of the difference in grade between two point samples is the same over the entire deposit, and that it depends only on the distance between and the orientation of the points. In other words, differences in grades must be consistent, *not* constant, over the deposit. This assumption is sometimes referred to as the intrinsic hypothesis or quasi-stationarity.

Now, consider the distribution of the difference in grade between two points *h* distance apart. Most distributions are described by their mean and variance, or standard deviation. In this case, the mean of distribution shows the change in expected grade between the two points. Thus, it reflects the trend in values over the deposit—the drift, in geostatistical terms. This mean is denoted $m(h)$, because it will depend, like the rest of the distribution, on *h*. Strictly speaking, *h* describes both distance and relative orientation, because the structure of the deposit may be different in different directions. If $m(h)$ is zero, there is no significant trend in the deposit.

For the purposes of estimation, the variance of the distribution is more useful than the mean. The variance, which also depends on *h*, should measure the interdependence of grades at two points separated by distance *h*. Normally, points close together will be more related than points farther apart. In geostatistical terms, if the points are highly related, then the variance of the distribution of differences will be low, and vice versa. Thus, this variance is a measure of the influence of samples over neighboring areas within the deposit. This variance is termed the variogram, since its usual representation is a graph of variance vs. the distance *h*. It is denoted mathematically by $2\gamma(h)$. The factor 2 is a matter of mathematical convenience. $\gamma(h)$ is called the half-variogram, or, more usually, the semivariogram, although many authors shorten this to variogram. It is always wise to check whether an author means γ or 2γ when using the term "variogram."

CONSTRUCTING A SEMIVARIOGRAM

The theoretical semivariogram γ is defined as one-half of the variance of the differences in sample grades between points separated by a distance *h*. It is theoretical in the sense that it is assumed to cover the whole deposit.

In practice, only a limited number of samples are available, and an experimental semivariogram must be constructed, usually called $\gamma^*(h)$ to distinguish it from the theoretical one. Throughout geostatistics, the asterisk is used to denote estimated or calculated values, as opposed to modeled or theoretically derived values.

From the above definition, constructing an experimental semivariogram from a set of data points would seem simple. However, to calculate a variance, one must first calculate a mean. In this case, the mean is the mean difference in

Fig. 1—Graphed semivariograms

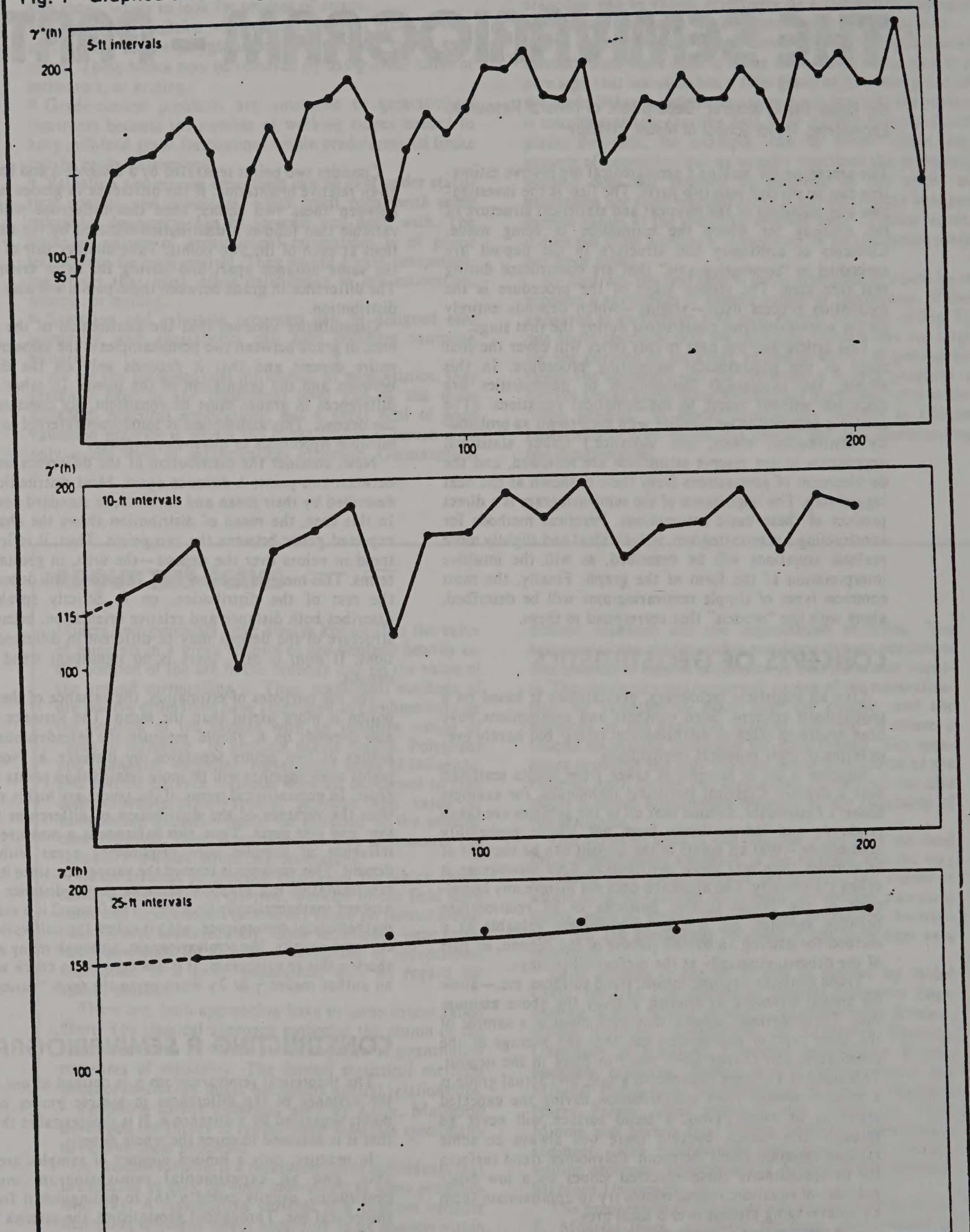
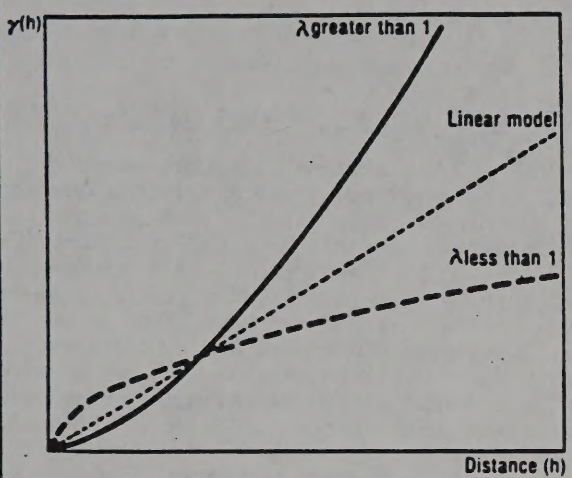


Fig. 2—Models without sills



grade—the drift or trend in values. Calculating the mean drift geostatistically is complicated and may not be necessary. Assume then that the trend in the deposit is negligible (a subsequent section will deal with cases in which this is not true). If the mean is zero, then the variance reduces to the average square, or in this case, the average of the squared differences in grades.

Suppose that point samples in a line are arranged regularly, a given distance apart. For simplicity, let this distance be called unit length and the values (grades, thicknesses, recovery, etc.) at these points g_1, g_2, g_3, and so on. The semivariogram must be calculated for various distances and plotted on a graph of γ^* vs. h. Obviously, values of γ^* can only be calculated at distances 1, 2, 3, etc., since these are multiples of the sampling interval. For distance 1, all pairs of points that distance apart are taken to calculate the semivariogram. The difference in grade is calculated and squared for each pair. The results are summed and divided by the number of pairs used. This result is halved. For example, if there are 20 samples:

$$\gamma^*(1) = \tfrac{1}{2} \times \tfrac{1}{19}[(g_1-g_2)^2 + (g_2-g_3)^2 + (g_3-g_4)^2 + \ldots + (g_{19}-g_{20})^2]$$

and for distance 2:

$$\gamma^*(2) = \tfrac{1}{2} \times \tfrac{1}{18}[(g_1-g_3)^2 + (g_2-g_4)^2 + \ldots + (g_{18}-g_{20})^2]$$

or in general:

$$\gamma^*(h) = \frac{1}{2(N-h)} \sum_{i=1}^{N-h} (g_i + g_{i+h})^2$$

for h 1, 2, 3 . . . 10 and N = 20.

Each of these values becomes one point on a graph of γ^* vs. h. As an illustration, consider an almost vertical cassiterite vein that has been developed by means of parallel, horizontal drives. Eighty-seven samples are taken at 5-ft intervals along the drive, and the grade is measured in pounds of black tin per ton of ore. The experimental semivariogram is calculated using the procedure described above. The result is the top graph in Fig. 1.

Note that although 430 ft of drive were sampled, the semivariogram has been calculated only for distances up to 215 ft. The theoretical restriction on distances between points used in γ^* is one-quarter of the total extent of the sample information, but in practice one-half is usually used. Note also that as the distance increases between pairs of points, fewer pairs go into the calculation. This implies that points on the graph closer to the origin (small distances) will be more reliable than those for greater distances, an important consideration when interpreting the semivariogram.

The semivariogram for the cassiterite vein in the example should give an indication of the structure of the deposit in terms of ore grade. The graph indicates that samples very close together—5 ft or closer—may be very different in grade value. If a line is projected through the first two calculated points, instead of joining the origin at zero it appears to cut the axis at 95 (lb per ton)². By definition, if assay error is ignored, the difference in grade between a point sample and itself must be zero. The graph implies that if two samples are taken very close together, a difference in grade of something close to 14 lb per ton can be expected. This is the nugget effect. It reflects to some extent the built-in random nature of mineralization of this type, a variation that cannot be predicted by any method.

If random variation accounts for a large proportion of the grade variation in a deposit, then a geostatistical reserve estimation might be of limited use.

The shape of the first graph in Fig. 1 rises and then falls, repeats this behavior, and then settles down, more or less, into a line that is reasonably straight, considering the small number of samples. This pattern, which suggests that at 40-ft and 80-ft intervals the difference in sample grade falls, appears because the ore occurs in shoots, and the drive crosses several of these shoots at a level where the average width of a shoot is about 40 ft. Pairs of samples taken on either side of a shoot will be more similar than pairs with one point in the middle and one on the edge. Even in this simple example with few samples, the geological structure of the deposit can be seen and quantified, without being masked by the highly erratic nature of the individual sample values.

The second and third graphs in Fig. 1 show the effect that a change of sampling interval can have on the interpretation of a semivariogram. The second graph shows the same calculated semivariogram with only every other point plotted as if samples were available only every 10 ft. The periodic structure is still visible and easily identifiable, but the "random" variation appears to have increased to 115 (lb per ton)², an increase in unpredictability of 20%.

The bottom graph shows points only at intervals of 25 ft. In this case, the deposit appears to have no structure whatsoever, with random variation from one sample to another. The implication of this example is not that samples should always be taken as close together as possible, but that the sampling interval should be borne in mind when interpreting the semivariogram.

In deposits that are essentially two-dimensional, such as vein deposits or sedimentary deposits of reasonably constant thickness, the ideal procedure is to collect samples on a square or rectangular grid. Several semivariograms can then be constructed, one in each of several directions. The semivariogram may be strongly influenced by the orientation of the points as well as the distance between them. Thus, it is necessary to construct, say, an east-west γ^*, a north-south γ^*, and perhaps a northeast to southwest and/or a northwest to southeast γ^*.

Then it must be decided whether the differences between

the semivariograms in the different directions represent real differences in the depositional nature of the ore or whether they are purely "statistical" variations resulting from a low number of samples. Semivariograms in different directions may give valuable information about continuity or the lack of it across a deposit. Perhaps an anisotropic structure is to be expected in alluvial deposits, lenticular deposits, and some vein-type deposits.

In practice, regular samples are rarely taken in all directions of interest. Most exploration programs are based on boreholes from which samples are taken at intervals of depth that are intended to be equal, but the boreholes are relatively great distances apart. Good semivariograms might be obtained down the hole, which is usually vertical, but information on the γ^* for the two horizontal directions will be sparse. Knowledge of the geology and structure of the deposit must be used to decide whether to assume that the deposit is isotropic. Even then, checks should be made as soon as development or adit samples are available, to adjust the semivariograms if necessary.

In a producing mine, samples are sometimes taken at regular time intervals that do not necessarily produce regular spatial intervals. The result is often a plan or section on which the sample points appear to be scattered arbitrarily. Obviously, in such a situation, construction of a semivariogram for one particular direction will be difficult. There will be only a small chance of finding more than a few pairs separated by exactly the same distance. To calculate γ^*, approximations have to be made. A direction is specified, and pairs are sought within "a few degrees" of that direction. Then a distance h is specified, and pairs are taken within some specified range to either side of that value for h. (This description is very general because actual practice depends on prevailing circumstances and is very much a rule of thumb. Experience and a thorough knowledge of the data obviously are necessary.)

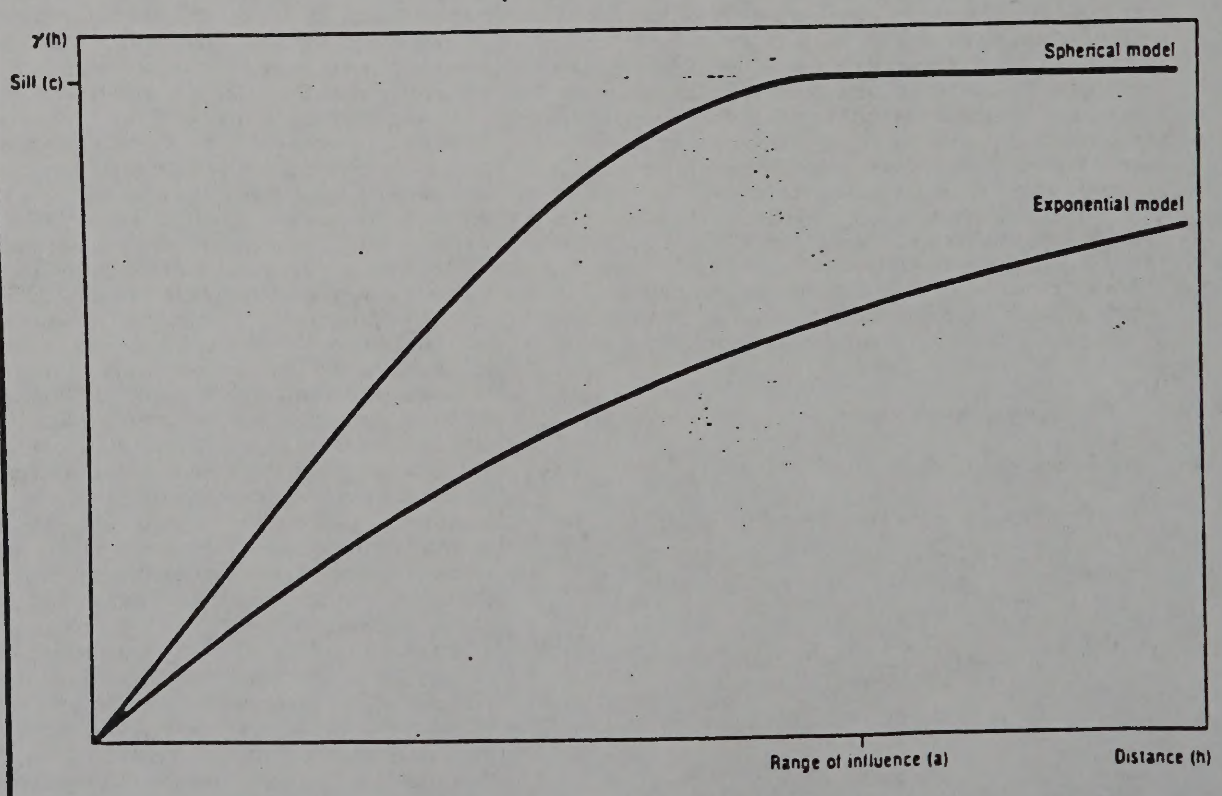
All in all, the construction of a semivariogram may be tedious and time-consuming, but it is rarely difficult.

SEMIVARIOGRAMS IN THEORY

While a calculated semivariogram may help in determining the structure of a deposit and the behavior of grade variations, it is purely a data summary technique—describing the behavior of the sample values. Conclusions about the whole deposit, if they are desired, must be produced by a process of inference. The process is analogous to the process of constructing a histogram from sample values and then inferring from the histrogram a theoretical distribution for the whole deposit. Certain distributions, such as normal or log-normal, if found, are more easily handled, and tables are available for use with them. Also, if an experimental semivariogram has been constructed, it must be related to some theoretical model if conclusions are to be drawn or estimations made for unsampled areas in the deposit. Such models are also used in kriging and other estimation procedures.

Just as there are few distributions available to the practi-

Fig. 3—Models with sills



CORRECTION

In the first article in this series, the paragraph that explains Fig. 5 (E&MJ, May, p 94) incorrectly specifies the ~~unit of measure~~ as deadweight tons. The correct unit is inch-pennyweight (in.-dwt); 1 dwt = 0.05 troy ounce.

The text should read: With a cutoff grade of 200 in.-dwt per block, the polygonal method indicates a reserve of 935 blocks having a mean grade of 530 in.-dwt per block. In fact, the true reserves at this cutoff grade are 1,400 blocks with a mean grade of 415 in.-dwt per block.

tioner of classical statistics, there are only a few simplistic models for a theoretical semivariogram. These divide into two groups, those in which the theoretical semivariogram $\gamma(h)$ increases as the distance h increases and those that increase at first and then tend to level off at a constant value of γ. The latter are said to have a sill, the value of which is usually represented by C.

Of the models without a sill, the most frequently used is the linear model. This semivariogram is simply a straight line passing through the origin of the graph and is defined by its slope p. The linear γ is shown as the dotted line in Fig. 2. A generalization of this model exists in which the value of γ is related to the distance h raised to some power λ that lies between zero and 2 but never actually equals 2. The other two curves in Fig. 2 show examples of the generalized linear model for specified values of λ. Only one other model is available without a sill—the De Wijsian model, which is essentially a straight line relationship between the γ values and the logarithm of the distance h.

There are three models that possess a sill, although one of them is mentioned very rarely. The two most commonly used are the spherical, or "Matheron," model and the exponential model. Fig. 3 shows the general shape of the two semivariograms for a specified sill C. Both models are virtually linear for small values of h, but the slope of the line is different. The spherical model rises rapidly, then gradually curves off until at a certain distance it reaches its sill and stays there. This distance is generally denoted by a and defines the distance within which samples may be assumed to bear some relationship to one another. Points farther apart than distance a are unrelated or independent of one another; therefore, a is referred to as the range of influence of a point within the deposit. The exponential model is also defined by these two parameters (sill and range of influence), but it rises more slowly than the spherical model and never quite reaches its sill. The two models in Fig. 3 have the same range of influence.

Models without sills

Linear	$\gamma(h) = ph$	
Generalized linear	$\gamma(h) = ph^{\lambda}$	$0<\lambda<2$
De Wijsian	$\gamma(h) = 3\alpha \log h$	

Models with sills

Exponential	$\gamma(h) = C[1-\exp(h/a)]$	
Spherical	$\gamma(h) = C\left[\dfrac{3h}{2a} - \dfrac{h^3}{2a^3}\right]$	$h<a$
	$= C$	$h>a$
Nugget effect	$\gamma(h) = 0$	$h = 0$
	$= C_0$	$h > 0$

The other model with a sill does not behave linearly near the origin but rather is parabolic. It then takes the same sort of shape as the spherical model in that it rises steeply toward the sill, but it reaches the sill in a smooth curve, rather than with the definite break of the spherical model. This model is said to be characteristic of data having a high degree of continuity, such as the thickness of some sedimentary deposits, and is called the Gauussian model.

One useful characteristic of models that have sills is that the value C should be equal to the ordinary variance of the sample values within the deposit. That is, if the grade (or any other variable) could be measured at every point in the deposit and then the variance of that set of samples calculated, the value would exactly equal C. Some authors take advantage of this behavior and define an "auto-covariogram" that is equal to the sill minus the semivariogram. In other words, they turn the semivariogram upside down, so that low values mean low interrelationships, etc. This does not alter the results or conclusions drawn from the analyses.

One other model of note is the one that represents purely random behavior. All samples are completely independent of one another, so the semivariogram consists only of a sill. This is the nugget effect mentioned previously. The value taken by the sill is always denoted by C_0. The γ value is equal to C_0 everywhere except when h is exactly zero, since $\gamma(0)$ must be zero by definition. The equations for all of these models are shown in the accompanying box. ■

REFERENCES

1) Huijbregts, C. J., "Regionalised Variables and Quantitative Analysis of Special Data," *Display and Analysis of Spatial Data*, Wiley-Interscience, 1975.

THE SEMIVARIOGRAM – PART 2

Dr. Isobel Clark, lecturer, Department of Mineral Resources Engineering, Royal School of Mines, London

The previous article in this series laid the theoretical foundations for a geostatistical ore reserve estimation. The existence and possible form of the semivariogram pertaining to any specific deposit were discussed in relation to this conceptual basis. However, before embarking upon the actual estimation of reserves, a relationship must be established between the practical, calculated semivariogram and the simplistic models. In practice, attempts to establish such a relationship may run into various difficulties. To aid the ore estimator, examples are given here of the most common deviations from the ideal and of how they are tackled. The effects of the size and shape of samples on the semivariogram are also discussed.

RELATING PRACTICE TO THEORY

Matching models and semivariograms is as fraught with practical difficulties as matching distributions and histograms. A good, stable experimental semivariogram with an easily recognizable sill may not look like either of the two models available—the spherical or the exponential. The assumption that the trend is negligible may well be erroneous. The deposit may be obviously anisotropic and require special attention. The curve may not go through the origin without bending drastically, requiring introduction of a nugget effect. It may appear that two or three of the simple models would have to be combined to produce a curve that would fit the experimental semivariogram.

In Fig. 1, for example, the solid line in the upper graph represents a γ^* calculated vertically down a collection of 43 boreholes scattered at random across a small uranium deposit. The lower graph shows the number of pairs of points that went into each calculated point on the semivariogram, giving an idea of the relative reliability of the γ^* values for large distances between samples. Little has been published on the sensitivity of the semivariogram to the number of pairs used in its construction, so such a graph is invaluable in conveying an essentially intuitive idea of stability. Rules of thumb for the number of pairs needed seem to be as numerous as the deposits to which geostatistics has been applied, with recommendations varying from 20-25 to 400 or 500.

In Fig. 1, there is obviously a very large nugget effect to be taken into account. If a line is projected through the first few data points, its intercept on the γ axis should serve as an estimate for C_0. Since the nugget-effect model is a horizontal line, the semivariogram becomes whatever model is chosen plus some constant C_0. Because the graph reaches a certain level and stays there, the spherical model must be chosen.

One feature of the spherical model is that if the line near the origin is taken and extended the other way, it will cut the sill when the distance h equals $2a/3$. If this is done on this curve, two lines meet at about 6½ m, giving a range of influence of just under 10 m. However, it is apparent that the curve does not reach its sill until something like 30 m. The smooth dotted line shows the model that was finally chosen after a process of trial and error. It is actually a mixture of two spherical models and a nugget effect. One model has a range of influence of about 4 m and a sill that has been denoted by C_1 and the other a range of 30 m and a sill of C_2.

A polynomial equation could have been produced that would describe the line and that would fit the γ^* points just as well, except for two reasons. First, in order to estimate reserves, a model is needed that can be handled with relative ease. If a new model is created for each semivariogram, all of the multiple integrals and/or computer approximations that go with it must also be created. Second, in this and many other cases, there are physical reasons for expecting differing degrees of continuity at different scales of investigation.

In the case of a uranium deposit, a short-range reinforcement of the radiation levels from sample to sample would be suspected, along with a longer-range underlying continuity in the actual individual grades and a high degree of random "noise." Mixed models have also been observed in other uranium deposits by Guarascio,[1] in copper vein deposits by Sinclair and Daraisme,[3] and by the author in a disseminated nickel deposit, where the model was supported completely by the company geologist's qualitative interpretation.

Fitting a model to an experimental semivariogram can also be difficult if the deposit is anisotropic or zoned. Ideally, kriging is done on the assumption that the deposit is isotropic. If it is not, then corrections must be applied so that for all intents and purposes it becomes isotropic. The most common type of anisotropy, and the easiest to handle, occurs with semivariograms in different directions exhibiting different ranges of influence. Alluvial deposits have a higher range of influence in the downstream direction than across or vertically through the flow. Lenticular deposits will probably have higher ranges along strike and downdip than across strike.

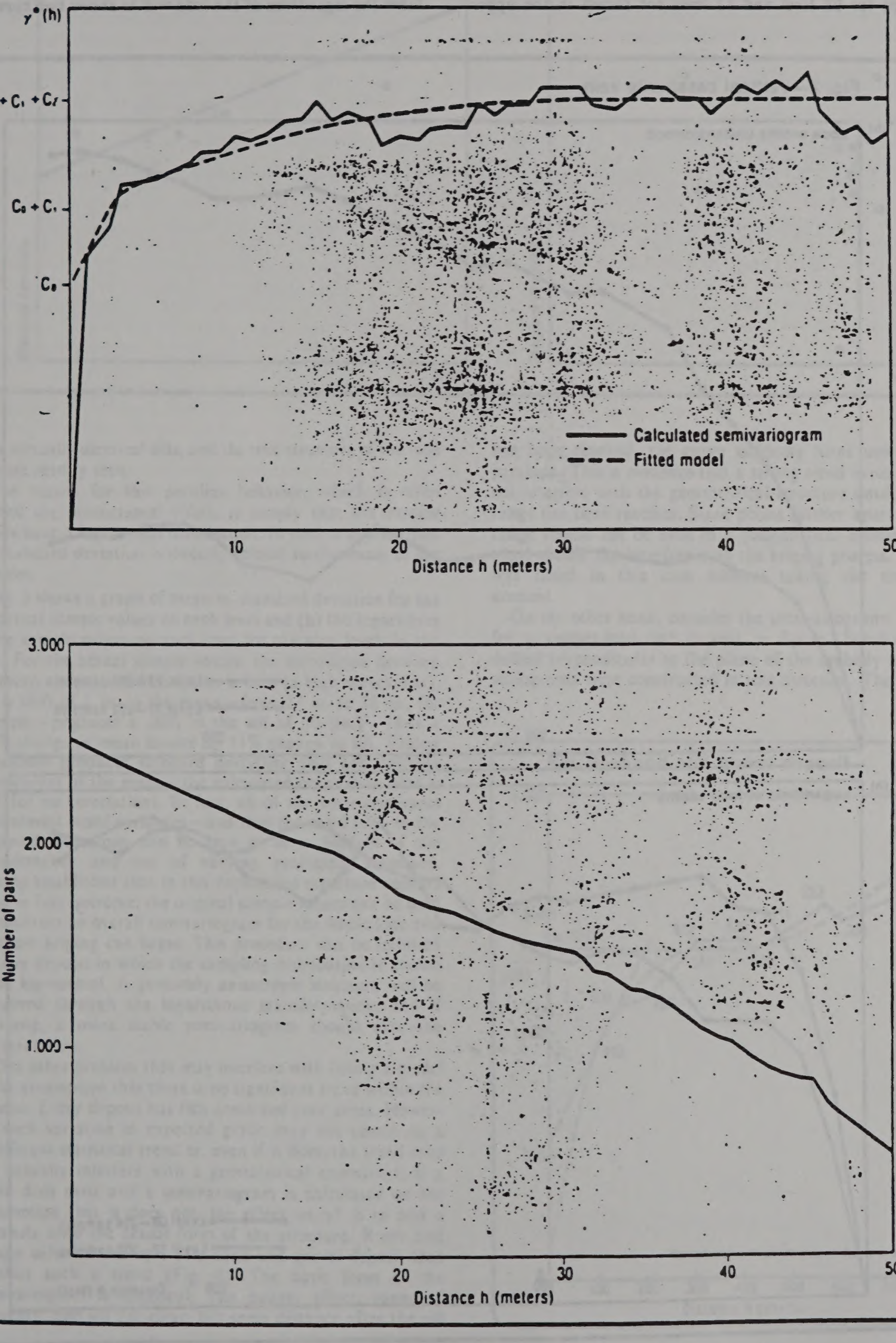
Base metal sulphides have been known to show distinctly shorter ranges in the vertical direction than in the two horizontal ones, as have some porphyry coppers. Such a problem is tackled by changing the units of measurement so that all ranges have the same numerical value.

For example, suppose a porphyry copper deposit has ranges in the two horizontal directions equal to 400 ft, while in the vertical direction the range is only 100 ft. If the units are changed in the vertical direction to ¼ ft, then the range becomes 400 units, which makes the deposit isotropic. Kriging is then carried out, bearing in mind that the unit in the vertical direction is 3 in., not 1 ft. To estimate a block of 50 ft x 50 ft x 12 ft, units would be expressed at 50 x 50 x 48 and the block would be kriged as if the entire deposit had a range of 400 units in all directions.

The other kind of anisotropy appears as different sills for semivariograms calculated in different directions or over different areas of the orebody. It is important for the estimator to decide if the difference in sills results purely from sampling variation. Any two sets of samples from the same distribution will have different variances—but the reason is that a finite number of samples are taken, not that there is an actual difference between the two sets. The same is true for a semivariogram sill, which should equal the sample variance. This is especially true with log-normally distributed samples, where the apparent anisotropy can be eliminated completely by construction of semivariograms using the logarithm of the sample values.

For example, consider the two graphs in Fig. 2. Six development drives have been driven following an almost vertical cassiterite vein at 100-ft vertical intervals, between 800 ft below surface and 1,300 ft below surface—08 level to 13 level. Samples were taken every 10 ft along these drives.

Fig. 1—Uranium deposit, Greenland



measuring grade in pounds per ton and the width of the vein in inches. Semivariograms were constructed on the vein widths, separately for each level. The experimental semivariograms for 08 level and 12 level are shown in the upper graph in Fig. 2. One might think that the structure of the deposit varies widely over this vertical stretch of 500 ft. However, a plot of the same two levels with the γ^* calculated from the logarithms of the lode widths shows two curves that

Fig. 2—Vertical cassiterite vein

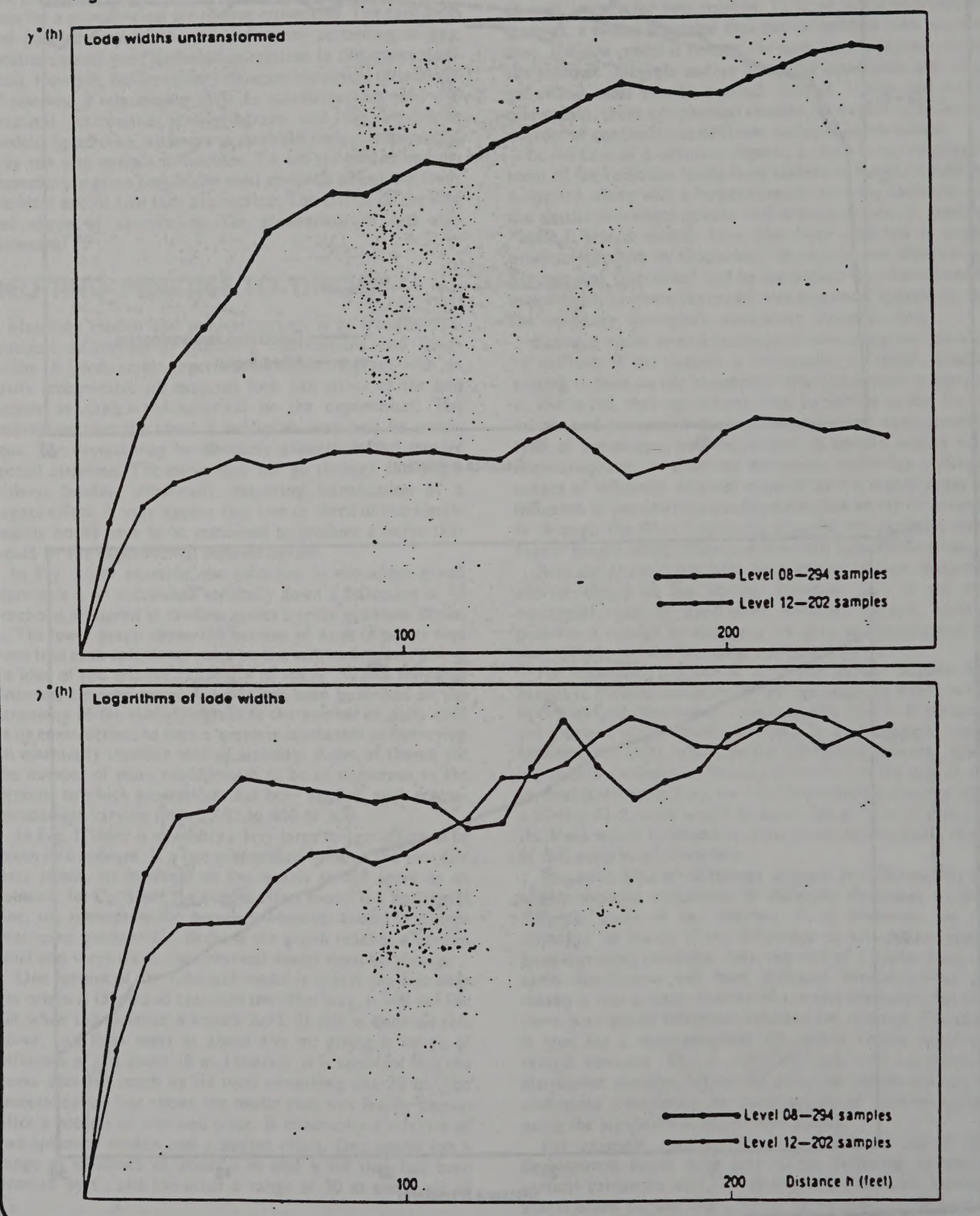
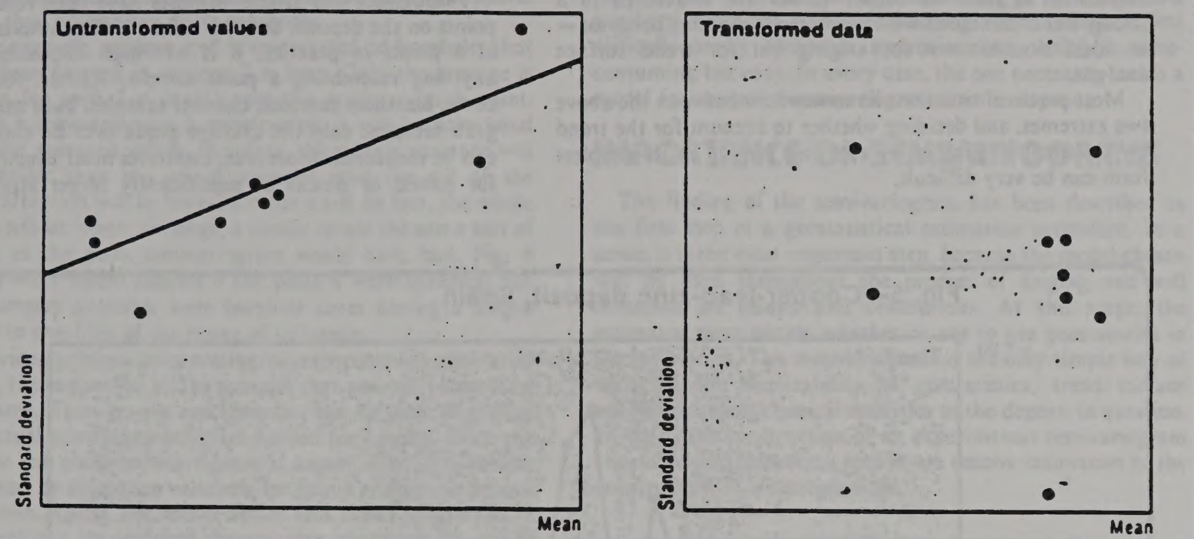


Fig. 3—Mean vs. standard deviation



have virtually identical sills, and the true structure of the lode is more readily seen.

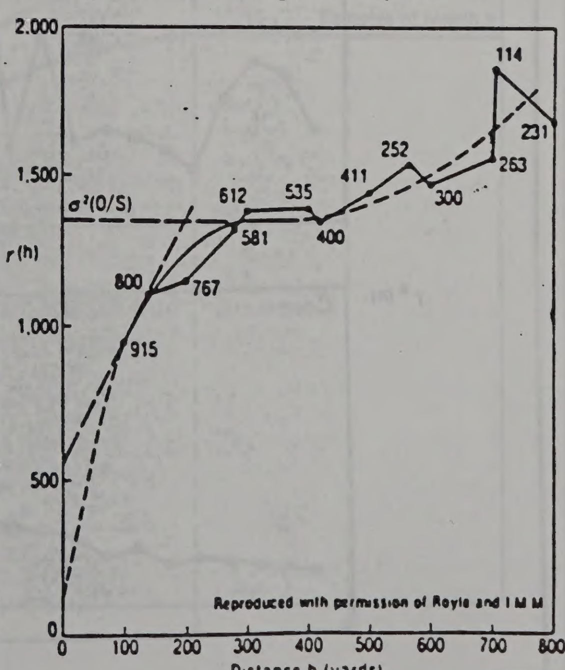
The reason for this peculiar behavior, which is often termed the proportional effect, is simply that the sample values have a log-normal distribution. In such a distribution, the standard deviation is directly related to the mean of the samples.

Fig. 3 shows a graph of mean vs. standard deviation for (a) the actual sample values on each level and (b) the logarithms of the sample values on each level for the nine levels in the lode. For the actual sample values, the correlation between the mean and standard deviation is 0.85 (a high correlation), and a shift of 1 in. in the mean—from 27 in. to 28 in., for example—produces a shift in the sill of 30 (in.)^2. Thus a 3½% change in mean causes an 11% change in sill. Taking logarithms produces standard deviations that are virtually independent of the mean—the calculated correlation here is 0.07 (or no correlation). In fact, all of the semivariograms have almost equal variances—and thus the same sill—so the supposed anisotropy can be seen to be a product of the log-normality and not of varying geological structure. Having established that in this deposit the structure appears to be in fact isotropic, the original sample values can be used to construct an overall semivariogram for the whole lode and isotropic kriging can begin. This procedure can be followed for any deposit in which the sampling distribution is known to be log-normal. A genuinely anisotropic structure will be preserved through the logarithmic transformation, and if anything, a more stable semivariogram should ease the interpretation.

One other problem that may interfere with fitting a model is the assumption that there is no significant trend within the deposit. Every deposit has rich areas and poor areas. However, such variation in expected grade may not constitute a significant statistical trend or, even if it does, the trend may not actually interfere with a geostatistical estimation. If a trend does exist and a semivariogram is calculated on the assumption that it does not, the effect on γ^* is to add a parabola onto the actual form of the structure. Royle and Hosgit offer an example of a sand and gravel deposit that exhibits such a trend (Fig. 4). The basic form of the semivariogram is spherical. The nugget effect, range of influence, and sill are clear, but some distance after the sill has been reached, the curve suddenly turns upward in a parabola. This is evidence that a strong trend exists but does not interfere with the geostatistical structure until after the range has been reached. Since points further apart than the range should not be used in a geostatistical estimation, the trend should not interfere with the kriging process. A model was fitted in this case without taking the trend into account.

On the other hand, consider the semivariograms in Fig. 5 for a copper-lead-zinc deposit in Spain. Boreholes were drilled perpendicular to the plane of the orebody and semivariograms were constructed in this direction. The lead and

Fig. 4—Sand and gravel deposit



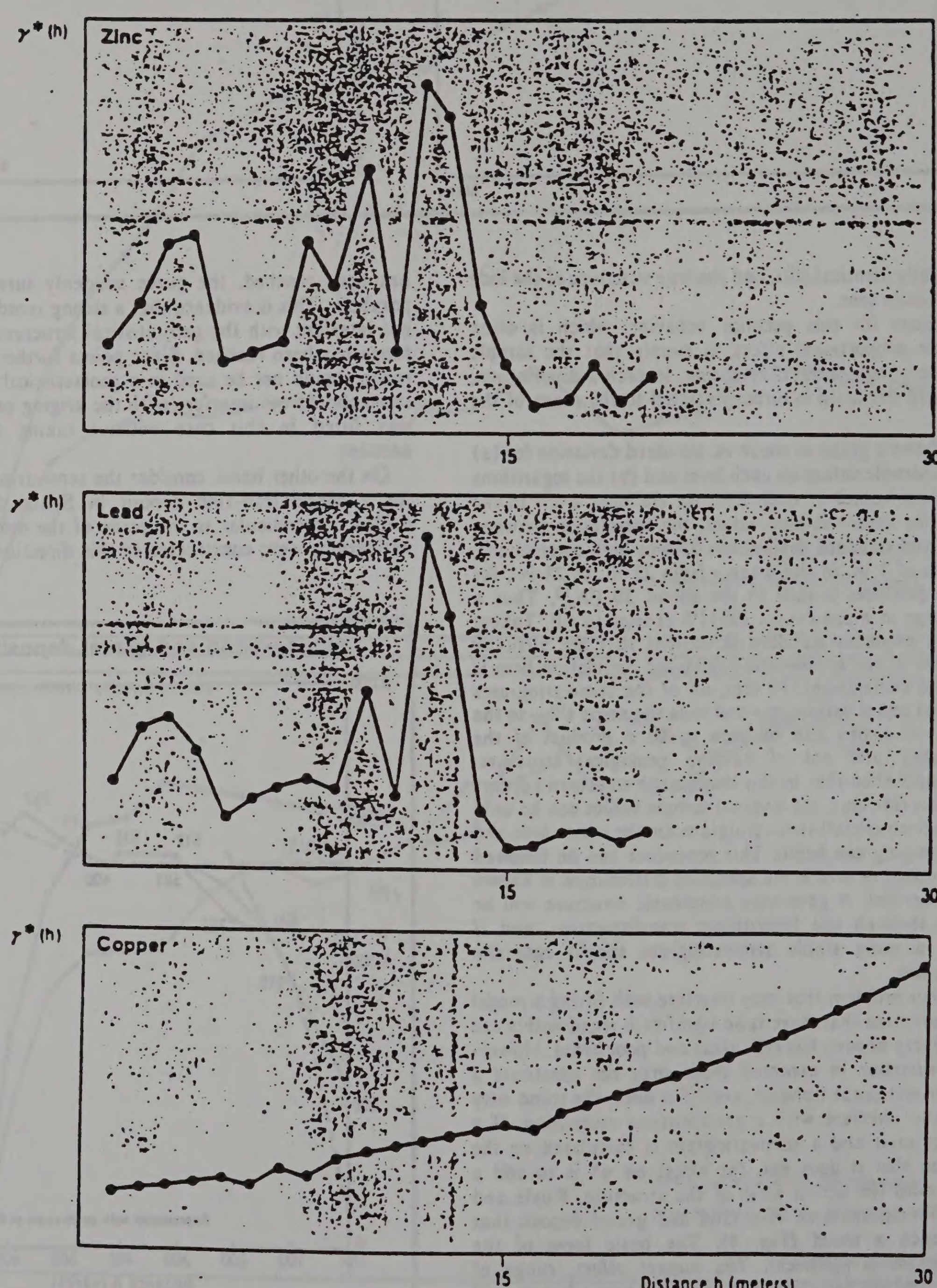
zinc values show nothing but nugget effect, while the copper values show a substantial nugget effect plus a parabola. The implication is that the copper values are structured in a strong trend combined with completely random behavior—an ideal situation not for kriging but for trend surface analysis.

Most practical situations lie somewhere between the above two extremes, and deciding whether to account for the trend by Universal Kriging or by applying kriging in its simplest form can be very difficult.

DEALING WITH REAL SAMPLES

Throughout this article, samples have been referred to as points on the deposit, and the values of the samples as values at a point. In practice, it is well-nigh impossible to take anything resembling a point sample. Rather, for borehole cores, blasthole samples, channel samples, bulk samples, and grab samples, only the average grade over an entire sample can be measured. Moreover, estimates must usually be made for panels or blocks of significantly larger size than the

Fig. 5—Copper-lead-zinc deposit, Spain



original samples. The size, geometry, weight, and method of analysis of a sample combine to form the support of the sample, and the semivariograms for samples having supports other than points may differ markedly.

For example, suppose a γ^* is constructed on boreholes that have been assayed in sections 2 m long. Only the average is known for each 2-m length, not the grade at each point. When a semivariogram is constructed, it will be calculated on these averaged values; therefore, the sample variance will be smaller than the actual variance, and the sill of the semivariogram will be lower, if it has a sill. In fact, the whole curve will be lower, although it should retain the same sort of shape as the point semivariogram would have had. Fig. 6 shows what would happen if the point γ were spherical but the samples available were borehole cores having a length equal to one-fifth of the range of influence.

Obviously, when constructing an experimental semivariogram, the support of all the samples that go into it should be the same. Then graphs and formulas can be used to extract the actual point semivariogram needed for kriging. Once the sill for the point semivariogram is known, then the sample variance, or dispersion variance, for points within the deposit is known. Using the model for γ and tables or graphs, a variance can be produced for any size, shape, or volume of ore required. (A subsequent article will illustrate the uses of this volume-variance relationship and its important role in planning.)

In addition to being able to derive the γ between samples of a specified support, one can obtain without too much difficulty the values of semivariograms between two volumes, for example, that may be of different supports. The kriging system requires the semivariogram between the samples (boreholes and the like) and the area or volume (blocks, panels, etc.) to be estimated. These can be found directly from γ by evaluating the theoretical integrals, which is tedious and difficult, or by calculating a few standard auxiliary functions and manipulating them, which is less difficult, or by computer approximation, which is time-consuming but easy. In every case, the one necessary tool is a model for the point semivariogram for the deposit.

IMPORTANCE OF SEMIVARIOGRAMS

The finding of the semivariogram has been described as the first step in a geostatistical estimation procedure. In a sense, it is the most important step, because the model chosen will be used throughout the process of kriging and will influence all results and conclusions. At this stage, the estimator must decide whether or not to use geostatistics in the estimation. The semivariogram is the only simple way of verifying the applicability of geostatistics, trend surface analysis, or even classical statistics to the deposit in question. In short, the construction of an experimental semivariogram should be as automatic a step in ore reserve estimation as the construction of a histogram.■

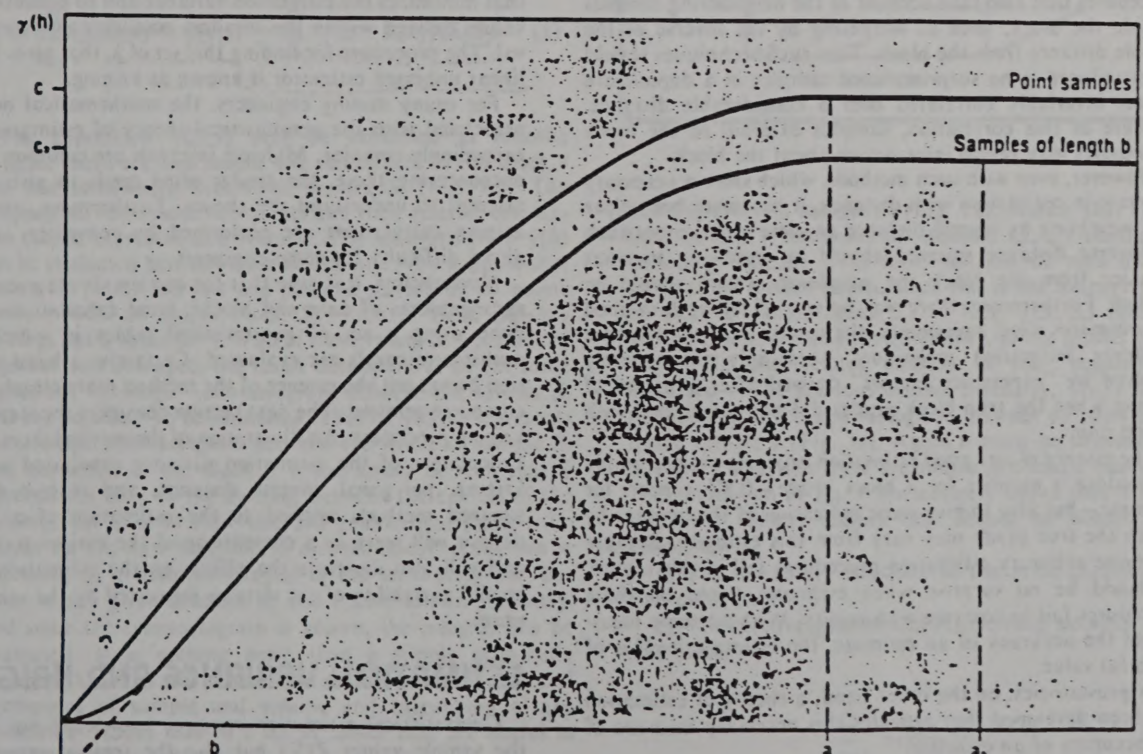
REFERENCES

1) Guarascio, M., "Improving the Uranium Deposits Estimations (The Novazza Case)," *Advanced Geostatistics in the Mining Industry*, Reidel, Holland, 1976.

2) Royle, A.G., and Hosgit, E., "Local Estimation of Sand and Gravel Reserves by Geostatistical Methods," Transactions of the Institute of Mining and Metallurgy, Vol. 83, April 1974, p A53-A62.

3) Sinclair, A. H., and Deraisme, J., "A Geostatistical Study of the Eagle Copper Vein, Northern British Columbia," CIM BULLETIN, Vol. 67, No. 746, p 131-142.

Fig. 6—Effect of sample "averaging"



KRIGING

Dr. P. I. Brooker, senior lecturer, Department of Economic Geology, University of Adelaide, South Australia

Commonly employed estimation techniques, such as inverse distance weighting, use a weighted average of sample values to estimate the mining blocks of a deposit. The weighting coefficients are a function of the block and sample geometry and of some general ideas of the variation in mineral deposits, but they make no reference to the particular variability of the orebody under study. Furthermore, these techniques do not allow a determination of the reliability of the estimates.

Geostatistical estimation techniques—based on a study of the spatial variability of the orebody as reflected in the semivariogram—are superior because they allow calculation of a measure of the error associated with the estimates, namely the variance of the error distribution. It is also possible to find the set of weighting coefficients for a given block and data configuration that minimizes this estimation variance. This procedure, which yields the best linear unbiased estimator, is known as kriging.

RELIABILITY OF ESTIMATES

Local estimation of the mining blocks of a deposit on the basis of available sample values is often carried out without regard for the variability of the orebody. The simplest approach, the polygonal method, which assigns a central sample value to the block, has sometimes been replaced by procedures that also take account of the neighboring samples outside the block, such as weighting by the inverse of the sample distance from the block. That such techniques should be introduced is no surprise, since samples in a deposit are almost invariably correlated over a considerable distance. Because of this correlation, samples external to the block boundaries may reveal information about the block.

However, even with such methods, which show a necessary decrease in correlation with distance, it has never been clear why weighting by inverse distance or some other factor such as inverse distance squared should be used—or to what distance from the block the weighting system should be applied. Furthermore, there was no way of telling how good the estimates were. Frequently, the results from these rather arbitrary estimation techniques were only inadequately resolved by "experience factors" derived after a period of mining when the true block grades did not correspond with expectations.

The essence of any good estimation procedure is not simply to produce a number for a block grade—0.8% copper, for example—but also to give some indication of the amount by which the true grade may vary from this estimate. Because the more arbitrary estimation procedures fail in this regard, it should be no surprise when estimates based on these procedures fail to coincide with reality. Without some measure of the accuracy of an estimate, the estimate itself is of doubtful value.

In geostatistics, on the other hand, a theory of estimation has been developed that provides this necessary measure of the accuracy of an estimate.

Whenever an estimate Z_v^* of a block of volume v whose true value is Z_v is made, an error $Z_v - Z_v^*$ is committed. It is, of course, impossible to know this error. If it were possible, there would be no estimation problems, as addition of the error and the estimate would give the true value:

$$(Z_v - Z_v^*) + Z_v^* = Z_v$$

In reality, there will be a distribution of the errors associated with the estimation of blocks over the deposit. Geostatistical theory shows that the variance of this distribution can be calculated in terms of the semivariogram. The variance of the distribution, whose mean value is usually forced to zero to avoid bias in the estimates, is a measure of the spread of the error distribution and thus gives an idea of the efficiency of the estimate. A large estimation variance relative to the estimate implies a poor estimate. The probability of such an estimate being far from reality is high. Conversely, a low estimation variance indicates an estimate close to reality. Frequently, a normal distribution is assumed for the error distribution, and standard confidence intervals are then defined for the block values.

All the previously mentioned estimation techniques use weighted averages of sample values for the block value: $Z_v^* = \Sigma_i \lambda_i Z(S_i)$, where λ_i is the weighting coefficient associated with sample S_i whose value is $Z(S_i)$. For the polygonal method, there is only the central sample, and $Z_v = Z(S_i)$; for the inverse distance methods, the λ_i are determined by the distance of the samples for the block. For a given block and data configuration, there is a set of weighting coefficients that minimizes the estimation variance and so produces block values defined within the smallest possible confidence interval. The procedure for finding this set of λ_i that gives the best linear unbiased estimator is known as kriging.

For many mining engineers, the mathematical notations associated with the geostatistical theory of estimation seem exceedingly complex. Multiple integrals are common and, on encountering these, the reader often tends to give up any attempt to understand the theory. Furthermore, almost all kriging calculations are performed by computer, which is also a difficulty for some engineers.

Nevertheless, it is true that for certain simple geometrical arrangements of data and blocks, hand calculations can be done using a set of geostatistical tables in which some multiple integrals are evaluated. Certainly, a hand calculation brings out the essence of the method more clearly than a computer printout. The next section considers the steps of the kriging procedure. An illustration of the method then follows. Calculation of the estimation variance associated with the kriging, polygonal, inverse distance, and inverse distance squared methods applied to the estimation of a tabular deposit will serve as a comparison of the various techniques and will also illustrate the effects on the estimation of the spatial variability of the data as measured by the semivariogram.

ESTIMATION VARIANCE AND KRIGING

Geostatistical estimation requires knowledge not only of the sample values $Z(S_i)$ but also the semivariogram $\gamma(h)$, which describe the spatial variability of the samples. Whether this semivariogram is isotropic or shows preferential directions of variability is of no account in developing the

estimation theory. The only requirements are that the blocks to be estimated lie within the neighborhood for which the semivariogram has been calculated, that enough samples be included to allow the experimental semivariogram to give a reliable picture of the variability in the neighborhood, and that the neighborhood be of homogeneous mineralization with no distinct trends in the data values. (The latter condition is the so-called quasistationary hypothesis mentioned in the previous article in this series.)

A deposit can often be divided into sections in which the above conditions apply. The blocks of each neighborhood are then kriged using their corresponding semivariogram.

If there is a trend in the mineralization at the scale of the size of the blocks, the situation is said to be nonstationary, and extensions of the kriging procedure must be applied. The literature covers this procedure under the heading "universal kriging." In this article, however, it is assumed that the conditions that apply will allow simple kriging. The positions of the block to be estimated and of the samples used in the estimation are incorporated into the kriging procedure. If the data are not on a regular grid, the block-data configuration will differ for each block of the deposit, and a strict kriging of the deposit, block by block, often will be impractical. An approximation technique known as random kriging has been developed to deal with such cases. (It will be discussed in a subsequent article in this series.)

The variance of the error associated with estimating blocks of true, unknown grade Z_v by a weighted average of n sample values:

$$Z_v^* = \sum_{i=1}^{n} \lambda_i Z(S_i),$$

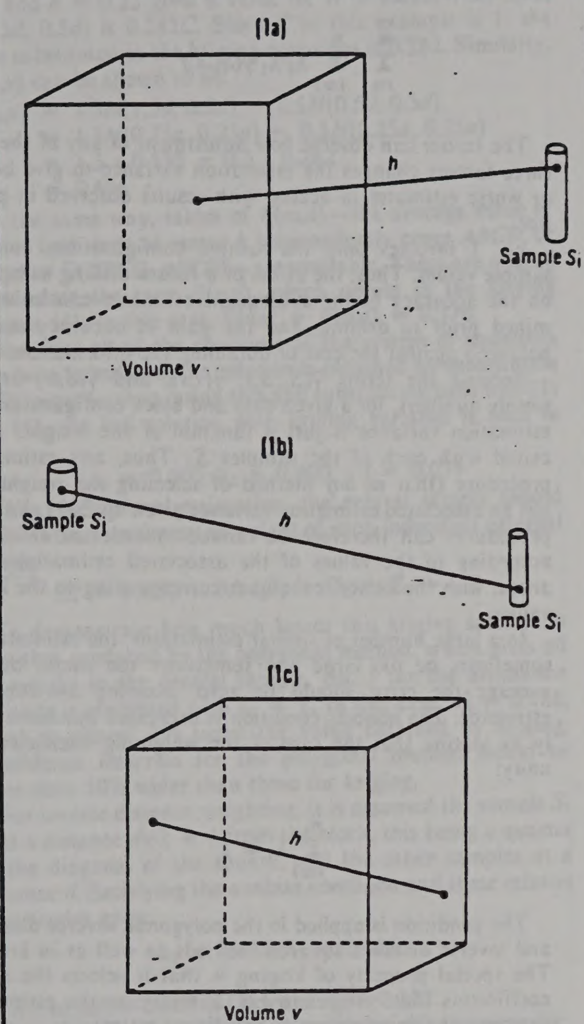
is written in terms of the semivariogram as

$$\sigma_E^2 = 2\sum_{i=1}^{n} \lambda_i \bar{\gamma}(S_i,v) - \bar{\gamma}(v,v) - \sum_{i=1}^{n}\sum_{j=1}^{n} \lambda_i\lambda_j\bar{\gamma}(S_i,S_j) \qquad (1)$$

The expression $\bar{\gamma}(S_i,v)$ may be explained as follows: Suppose there are two points, one somewhere in the block v and one somewhere within the sample S_i (Fig. 1a). The distance between these two points and their relative orientation can be expressed as a vector h. The semivariogram $\gamma(h)$ can be evaluated between these two points. Now suppose that these two points are just one possible pairing of all points inside v with all points inside S_i. Each of these possible pairings has a value for γ. If all of these values of γ are added together and averaged, the result obtained will be (in some sense) the "average" semivariogram between the sample S_i and the volume v to be estimated. That is, $\bar{\gamma}(S_i,v)$ represents the average semivariogram between the whole of sample S_i and the whole of the block v. It can easily be seen that $\bar{\gamma}$ depends not only on the distance between the centers of the volumes but also on their size and geometrical shape—the reason for the new notation. In mathematical terms, this is expressed as a multiple integral of the semivariogram, but since in any given situation S_i and v have known geometry, and since the semivariogram is known, the integral can be evaluated. It is nothing more than a simple arithmetic number. (Later, it will be seen how definition of a few special geometries of sample and volume and calculation of the associated $\bar{\gamma}(S_i,v)$ lead to a set of tables that are useful in evaluating other geometries.)

In the same way, $\bar{\gamma}(S_i,S_j)$ is the average semivariogram between all pairs of points, where one point is in sample S_i and the other in sample S_j (Fig. 1b). Notice that the third term in Eq. 1 comprises all possible (S_i,S_j) combinations, including $\bar{\gamma}(S_i,S_i)$. This term will not be zero unless the samples are simple points, since this is the average between *all* pairs of points inside S_i. In fact, $\bar{\gamma}(S_i,S_i)$ reflects the differences, and hence the variability, of the grades of points within S_i, and relates directly to the "regularization" discussed in the previous article. Similarly, $\bar{\gamma}(v,v)$ is simply the average of the semivariogram between all pairs of points inside volume v (Fig. 1c). Again it must be stressed that in any given situation these are simple arithmetic numbers.

Fig. 1—Significance of the terms in equation 1



Examination of Eq. 1 immediately shows that it incorporates the various factors that should be heeded in any estimation:

1) The variability structure of the mineralization: all of the terms involve $\bar{\gamma}$.

2) The relative geometrical relationship between samples and block:

$$\sum_{i=1}^{n} \lambda_i\bar{\gamma}(S_i,v)$$

3) The geometry of the block to be estimated: $\bar{\gamma}(v,v)$.

4) The configuration of the samples:

$$\sum_{i=1}^{n}\sum_{j=1}^{n}\lambda_i\lambda_j\bar{\gamma}(S_i,S_j)$$

The reader can observe how adjustment of any of the last three factors changes the estimation variance to give better or worse estimates in accord with results observed in practice.

Eq. 1 involves only the sample configuration, not the sample values. Thus, the effect of a future drilling campaign on the accuracy of the subsequent estimates can be determined prior to drilling, and the gain in accuracy can be balanced against the cost of obtaining the information.

Because the terms $\bar{\gamma}(S_i,S_j)$, $\bar{\gamma}(v,v)$, and $\bar{\gamma}(S_i,v)$ are all simply numbers, for a given data and block configuration the estimation variance is just a function of the weights associated with each of the samples S_i. Thus, any estimation procedure (that is, *any* method of selecting the weights λ_i) has an associated estimation variance given by Eq. 1. Various procedures can therefore be ranked in order of efficiency according to the values of the associated estimation variances, with the better techniques corresponding to the lower values.

In a large number of similar estimations, the estimate will sometimes be too large and sometimes too small, but on average the error should be zero, assuring an unbiased estimator. The nonbias condition is expressed mathematically by stating that the sum of the weighting coefficients is unity:

$$\sum_{i=1}^{n}\lambda_i = 1 \tag{2}$$

The condition is applied in the polygonal, inverse distance, and inverse distance squared methods as well as in kriging. The special property of kriging is that it selects the set of coefficients that, subject to Eq. 2, minimize the estimation variance and so produce the best linear estimator.

This minimization is brought about by solving a set of linear equations, which for two samples S_1 and S_2 (as in the practical example detailed below) is:

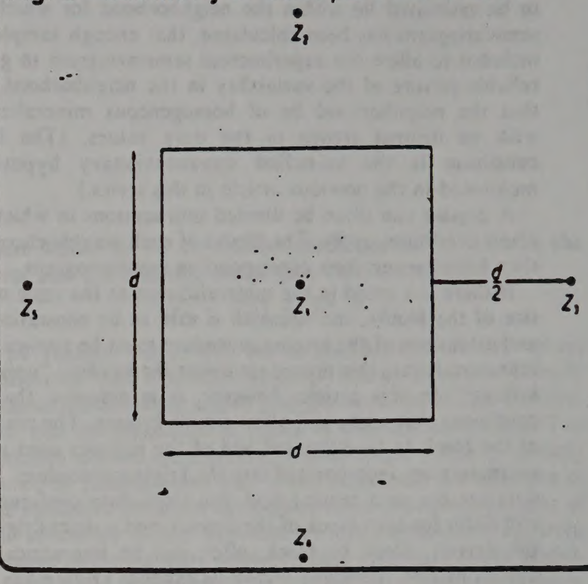
$$\begin{aligned}\bar{\gamma}(S_1,S_1)\lambda_1 + \bar{\gamma}(S_1,S_2)\lambda_2 + \mu &= \bar{\gamma}(S_1,v)\\ \bar{\gamma}(S_2,S_1)\lambda_1 + \bar{\gamma}(S_2,S_2)\lambda_2 + \mu &= \bar{\gamma}(S_2,v)\\ \lambda_1 + \lambda_2 &= 1\end{aligned} \tag{3}$$

Recalling that the $\bar{\gamma}$ are just numbers, it can be seen that there are three equations in three unknowns, λ_1, λ_2, and μ, the latter being an unknown constant introduced in the minimization process. There is no difficulty in solving such a small system of equations by hand, and using the values for λ_1, λ_2, and μ, the minimum estimation variance can be obtained from the formula:

$$\sigma_k^2 = \sum_{i=1}^{n}\lambda_i\bar{\gamma}(S_i,v) + \mu - \bar{\gamma}(v,v) \tag{4}$$

(In the event of a larger number of n samples S_i, the $n+1$ equations in $n+1$ unknowns, λ_1, λ_2, ... λ_n, μ, which are an immediate generalization of the set (3), are solved by computer.)

Fig. 2—Geometry of example cited in text



PRACTICAL EXAMPLE

Suppose that a horizontal tabular deposit is sampled by vertical drillholes located on a regular grid. In such a situation, the valuation can often be reduced to a two-dimensional problem, where the variables considered are accumulations (average grade over the mineralized thickness, times thickness), and the spatial variability of such variables is studied in the horizontal plane. For the purposes of this exercise, the tabular body will be evaluated using square blocks of side d, which will be estimated with five samples. Z_1 is at the center of the block, and Z_2, Z_3, Z_4, and Z_5 are at the centers of the adjoining blocks, as shown in Fig. 2.

The semivariogram of the deposit will be assumed to be isotropic and of the spherical type, with a range $a = 2d$, nugget effect $C_0 = 0$, and sill $C_0 + C = 1$.

Because of the symmetry of samples Z_2, Z_3, Z_4, and Z_5 and because the semivariogram shows the same variability in all directions, each of these four samples will receive the same weight. This fact (which can easily be demonstrated by writing down the 6 x 6 kriging system, treating each of the samples separately) allows these samples to be grouped together to form the information set S_2. The estimate for the block will then be:

$$Z_v^* = \lambda_1 Z(S_1) + \lambda_2 Z(S_2)$$

where

$Z(S_1) = Z_1$

$Z(S_2) = 0.25\,(Z_2 + Z_3 + Z_4 + Z_5)$: the value associated with the grouped set.

Now, since there are only two coefficients, λ_1 and λ_2, it follows from the nonbias condition (2) that $\lambda_2 = 1 - \lambda_1$, so that the kriging system of equations can be written:

$$\bar{\gamma}(S_1,S_1)\,\lambda_1 + \bar{\gamma}(S_1,S_2)\,(1-\lambda_1) + \mu = \bar{\gamma}(S_1,v)$$

$$\bar{\gamma}(S_2,S_1)\,\lambda_1 + \bar{\gamma}(S_2,S_2)\,(1-\lambda_1) + \mu = \bar{\gamma}(S_2,v)$$

Subtraction of the first of these from the second eliminates μ and gives:

$$\lambda_1 = \frac{\bar{\gamma}(S_2,v) - \bar{\gamma}(S_1,v) + \bar{\gamma}(S_1,S_2) - \bar{\gamma}(S_2,S_2)}{2\bar{\gamma}(S_1,S_2) - \bar{\gamma}(S_1,S_1) - \bar{\gamma}(S_2,S_2)} \tag{5}$$

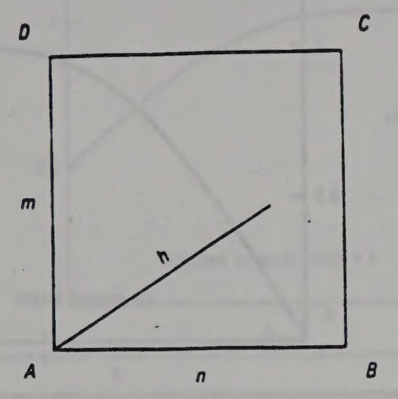


Fig. 3—$H(m, n) = \bar{\gamma}(A, ABCD)$

Thus, all that remains is to evaluate the $\bar{\gamma}$, the coefficients of the equations. The terms on the left hand side of the equations involve only the sample sets S_1 and S_2 and are:

$$\bar{\gamma}(S_1, S_1) = 0$$

$$\bar{\gamma}(S_1, S_2) = \bar{\gamma}(S_2, S_1) = \gamma(d) = 0.688$$

$$\bar{\gamma}(S_2, S_2) = 0.25[2\gamma(d\sqrt{2}) + \gamma(2d)] = 0.692$$

these results being obtained from the formula for the spherical semivariogram.

The terms $\bar{\gamma}(S_1,v)$ and $\bar{\gamma}(S_2,v)$ can be written in terms of a so-called auxiliary function, which is defined for a particular geometry of block and sample thus: $H(m,n)$ is the average value of $\gamma(h)$ when one end of the vector h is fixed at A, one corner of the rectangle $ABCD$, and the other covers the area mn of the rectangle (see Fig. 3). This function can be tabulated for any spherical model,[1] but it is usual to refer to a standardized table that can be converted to the required sill and range of influence.

In a spherical model, all distances are expressed as being relative to the range of influence a. This means that the value of $H(m,n)$, for example, for a model with range a, is the same as the value of $H(m/a, n/a)$ for a model with range 1. Thus, a table with range 1 covers *all* spherical models. Similarly, in these tables the sill is set to 1. To obtain values for a sill C, simply multiply the tabled value by C. Table 1 shows a selection of $H(m,n)$ values for a spherical model with $a = C = 1$.

To return to the example above, it can be shown that the term $\bar{\gamma}(S_1,v)$ is equal to $H(0.5d, 0.5d)$ where $d = a + 2$. The term that is required is therefore $H(0.25a, 0.25a)$ with the range a, or $H(0.25, 0.25)$ with range 1. From Table 1, $m = 0.25$ and $n = 0.25$ gives a value for H of 0.282. Therefore, $H(0.5d, 0.5d)$ is $0.282C$. Since C in this example is 1, the value substituted in the kriging equations is 0.282. Similarly, $\bar{\gamma}(S_2,v)$ can be shown to be:

$$\begin{aligned}\bar{\gamma}(S_2,v) &= 1.5H(1.5d, 0.5d) - 0.5H(0.5d, 0.5d)\\ &= 1.5H(0.75a, 0.25a) - 0.5H(0.25a, 0.25a)\\ &= 1.5 \times 0.558 - 0.5 \times 0.282\\ &= 0.696\end{aligned}$$

In the same way, tables of $F(m,n)$—the average value of $\gamma(h)$ as both ends of vector h independently cover $ABCD$, or $F(m,n) = \bar{\gamma}(ABCD, ABCD)$—are available. These are needed to calculate the term $\bar{\gamma}(v,v)$, which occurs in the kriging variance (4). In this case, $\bar{\gamma}(v,v) = F(d,d) = 0.376$.

Now that all of the constants of the system of equations have been calculated, the solution is obtained by substitution in (5) to get λ_1; then, using this and substituting back into (3) and (4), the full solution with kriging variance is seen to be:

$$\lambda_1 = 0.600,\ \lambda_2 = 0.400,\ \sigma_k^2 = 0.079$$

Thus, for optimal estimation, the central sample should receive about six times the weight of each individual external sample:

$$Z_v^* = 0.6Z_1 + 0.1\ (Z_2 + Z_3 + Z_4 + Z_5)$$

To demonstrate how much better this kriging solution is than that produced by the polygonal method, which gives all the weight to the central sample, Eq. 1 for the estimation variance is evaluated with $\lambda_1 = 1$. In this case, $\sigma_E^2 = 0.188$, which is almost 2½ times the value obtained by kriging. Confidence intervals for the polygonal method would be more than 50% wider than those for kriging.

For inverse distance weighting, it is assumed the sample S_1 is at a distance $d\sqrt{2} \div 4$ from the block, this being a quarter of the diagonal of the square, and the other samples at a distance d. Satisfying the nonbias condition and these relative magnitudes gives:

$$\lambda_1 = 0.414,\ \lambda_2 = 0.586$$

which on substitution in (1) yield $\sigma_E^2 = 0.102$.

The inverse distance squared method yields these results:

$$\lambda_1 = 0.667,\ \lambda_2 = 0.333,\ \sigma_E^2 = 0.082$$

Thus in this case, where the mineralization is continuous (nugget effect = 0), and for this block and data configuration, the inverse distance squared method is almost optimal and the inverse distance method not much worse. Certainly, if the weighting coefficients of a given technique vary little from the optimal kriging weights, the corresponding estimation variance will be near the kriging variance. However, this can only be determined by a geostatistical study.

To illustrate the effects of changes in the variability of the orebody on the estimation variance, the same calculations have been repeated for different spherical semivariograms. All variograms are kept at the same sill value, $C_o + C = 1$, but the relative nugget effect C_o/C and the range a are altered. For each of the three ranges $a = d$, $a = 2d$, and $a = 10d$, the nugget effect C_o is taken as 0 and 0.5. Finally, a pure nugget effect with $C_o = 1$ is studied. Fig. 4 shows the two semivariograms studied for a particular range and that of the pure nugget effect. These reflect mineralization that passes from quite continuous to very erratic. The results are given in Table 2, the parameter $R = \sigma_E / \sigma_k$ being the ratio of the normal confidence interval of the particular technique to that for kriging.

Examination of Table 2 shows that the kriging weights change according to the semivariogram of the deposit, whereas the weights in the other methods are determined once and

Table 1—H(m,n) for spherical model, a = C = 1

	m			
n	0.05	0.15	0.25	0.50
0.05	0.057	0.123	0.193	0.364
0.15	0.123	0.171	0.231	0.389
0.25	0.193	0.231	0.282	0.425
0.50	0.364	0.389	0.425	0.535
0.75	0.513	0.531	0.558	0.644
1.00	0.628	0.642	0.662	0.728
1.50	0.752	0.761	0.775	0.819

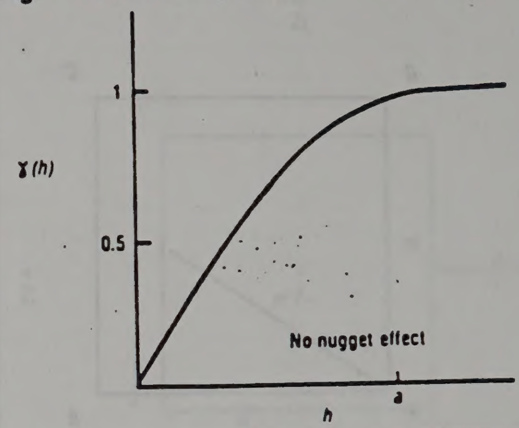
for all by the relative geometry of the samples and the block. In kriging, for a given range, the weight given the well-placed internal sample is always highest when the nugget effect is least; in generally continuous mineralization, the best-placed sample gets the most weight. As the nugget effect rises, mineralization becomes more erratic, and the placement of the samples becomes less important until, in the limit of a pure nugget effect, each sample finally receives the same weight. As the range increases, the external samples generally begin to appear better placed relative to the block and so receive more weight at the expense of the central sample. (The exception to this statement is the case of continuous mineralization, $C_o = 0$, and range equal to the block side, $a = d$, demonstrating the importance of the size of the block relative to the range and preventing absolute generalizations on the effect of increasing range.)

From this range of variograms and with this particular sample and block geometry, it is seen that the inverse distance squared method, which heavily weights the central sample relative to the external samples, is very nearly as good as kriging in situations where the mineralization is continuous. However, as the mineralization becomes more erratic, this estimator becomes increasingly less effective than kriging, until in the case of a pure nugget effect, the standard confidence interval is over 50% larger than for kriging.

On the other hand, the inverse distance method is almost as efficient as kriging over the whole range of semivariograms studied, while the polygonal method is inefficient relative to kriging.

Examples can be cited for which the inverse distance estimator is much worse than kriging. Indeed, for a highly anisotropic deposit, blind application of the inverse distance method would lead to inefficient estimates. Furthermore, a change in the data configuration relative to the block or a simple change in the number of samples used in the estimate can change the efficiency of the estimation technique relative to kriging. To illustrate this, consider the same block, $d \times d$, situated with its center at a grid point of a grid $2d \times 2d$, and again let the central sample and the four nearest neighbors be used in the estimation. The results for a spherical variogram with parameters $C_o = 0.5$, $C = 0.5$, and $a = 10d$ are given in Table 3.

Fig. 4—Semivariograms



Note that in the first situation, one set of kriging weights could be used for all the blocks of the deposit, since—ignoring blocks on the boundary—the block data configuration is repeated throughout. Here, however, there are three different configurations, when one considers all the blocks of size $d \times d$ estimated from a $2d \times 2d$ data grid. Thus, two other kriging systems, in addition to the one solved above, need to be solved.

As noted earlier, the weights in inverse distance and inverse distance squared methods are determined by the geometry of the samples relative to the block and are independent of the variogram. Additionally, these weights are determined without regard to the relative position of the samples, one to another.

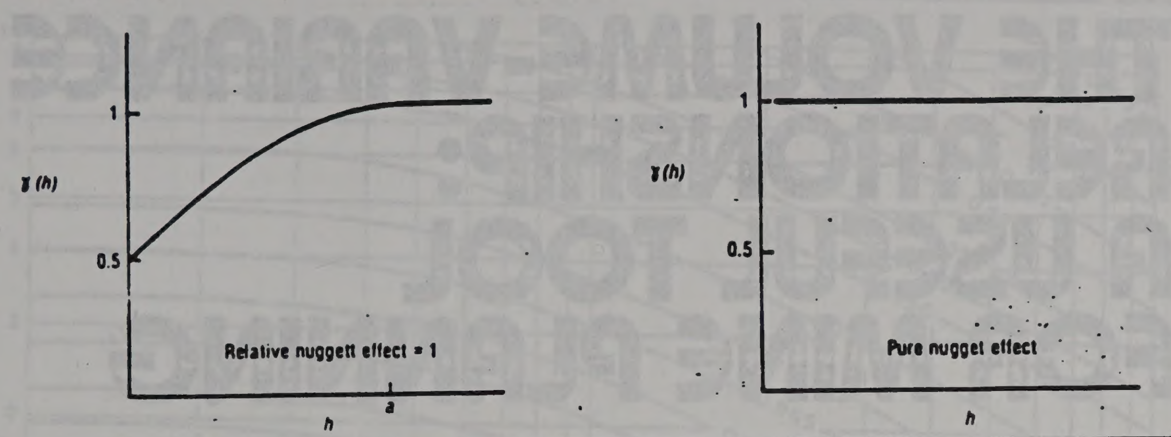
Suppose that in Fig. 2 the four external samples were in different positions relative to one another but not to the block, as in Fig. 5. Inverse distance gives the same weights to samples 2 and 5 and a very slightly smaller weight to each of samples 3 and 4 (see Table 4). However, the kriging system considers the actual "information" content of each sample. Samples 3 and 4 are so close together ($d/2$ units) that they each give information about essentially the same area. Thus,

Table 2—Comparison of estimation techniques for various spherical semivariograms

Nugget effect	Range	Kriging λ_1	Kriging λ_2	Kriging σ_k^2	Polygonal λ_1	Polygonal λ_2	Polygonal σ_E^2	Polygonal R	Inverse distance λ_1	Inverse distance λ_2	Inverse distance σ_E^2	Inverse distance R	Inverse distance squared λ_1	Inverse distance squared λ_2	Inverse distance squared σ_E^2	Inverse distance squared R
$C_o = 0$	$a = d$	0.541	0.459	0.144	1	0	0.407	1.68	0.414	0.586	0.164	1.07	0.667	0.333	0.164	1.07
	$a = 2d$	0.600	0.400	0.079	1	0	0.188	1.55	0.414	0.586	0.102	1.14	0.667	0.333	0.082	1.02
	$a = 10d$	0.573	0.427	0.015	1	0	0.037	1.57	0.414	0.586	0.018	1.10	0.667	0.333	0.016	1.03
$C_o = 0.5$	$a = d$	0.370	0.630	0.208	1	0	0.704	1.84	0.414	0.586	0.211	1.01	0.667	0.333	0.318	1.24
	$a = 2d$	0.341	0.659	0.175	1	0	0.594	1.84	0.414	0.586	0.180	1.01	0.667	0.333	0.277	1.28
	$a = 10d$	0.233	0.767	0.115	1	0	0.518	2.12	0.414	0.586	0.138	1.09	0.667	0.333	0.244	1.46
$C_o = 1$		0.2	0.8	0.2	1	0	1.0	2.24	0.414	0.586	0.257	1.13	0.667	0.333	0.472	1.54

Table 3—Estimation of block d x d by two centered grids
(for a spherical semivariogram with $C_o = 0.5$, $C = 0.5$, $a = 10d$)

Grid	Kriging λ_1	Kriging λ_2	Kriging σ_k^2	Polygonal λ_1	Polygonal λ_2	Polygonal σ_E^2	Polygonal R	Inverse distance λ_1	Inverse distance λ_2	Inverse distance σ_E^2	Inverse distance R	Inverse distance squared λ_1	Inverse distance squared λ_2	Inverse distance squared σ_E^2	Inverse distance squared R
$d \times d$	0.233	0.767	0.115	1	0	0.518	2.12	0.414	0.586	0.138	1.09	0.667	0.333	0.244	1.46
$2d \times 2d$	0.294	0.706	0.146	1	0	0.518	1.88	0.586	0.414	0.210	1.20	0.889	0.111	0.410	1.68



the two samples combined should, by common sense, only have about the same weight as sample 2. From Table 4, one can see that the kriging system automatically takes this into account and produces what appears to be a more sensible set of weights. The weight assigned the two adjacent points is considerably less than that allotted to either of the two outside the block. Also, the estimation variance shows that ignoring the relationship between the sample positions increases the confidence limit of the estimation by more than 25%.

The important point is that only with a geostatistical study, in which the variability of the ore is first characterized through the semivariogram, is it possible to make these calculations of estimation variance and so determine whether a specific estimation technique is satisfactory. In some instances, inverse distance and inverse distance squared methods might produce results close to those obtained by kriging, but the kriging results might indicate such a large estimation variance that further drilling is necessary to better define the orebody. The economic consequences of such action must then be evaluated. This can be done prior to any drilling, since the estimation variance can be evaluated for any proposed drilling grid.

THE KEY STEPS IN KRIGING

The kriging method of geostatistical estimation can be summarized as follows:

1) Structural study to determine the semivariogram.

2) Selection of samples to be used in evaluating the block.

3) Calculation of the $\bar{\gamma}$ of the kriging system of equations.

4) Solution of the system of equations to get the optimal weighting coefficients.

5) Use of these results to calculate the block estimate and associated estimation variance.

Certainly, this is a much more complex procedure than the application of a set of weighting coefficients of some arbitrarily chosen alternative technique. However, much effort has been devoted to developing computing systems for the kriging of orebodies which, from the point of view of costs, compete quite satisfactorily with other methods.[2]

As already noted, kriging should be applied only in homogeneous "quasistationary" neighborhoods for which the semivariogram is well defined. Throughout the orebody these semivariograms may change markedly, and it is most unlikely also that the drilling configuration will be consistent over the whole deposit. These factors affect the estimation variance, so an arbitrarily chosen estimator that is good in one area may be inadequate in another. Nevertheless, kriging guarantees the best result from the available data.

Fig. 5—Variation of geometry in Fig. 2

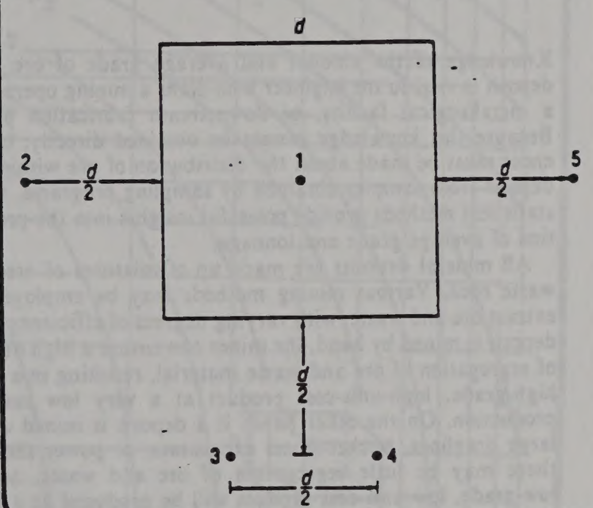


Table 4—Comparison between kriging and inverse distance for block data configuration of Fig. 5

(with a spherical semivariogram with $C_o = 0$, $a = 2d$, $C = 1$)

	Weight	
Point	Inv. dist.	Kriging
1	0.418	0.652
2	0.148	0.130
3	0.143	0.044
4	0.143	0.044
5	0.148	0.130
Est. Variance	0.152	0.095

REFERENCES

1) Clark, I., "Some Auxiliary Functions for the Spherical Model of Geostatistics," COMPUTERS AND GEO-SCIENCES, Vol. 1, 1976, pp 255-263.

2) David, M., "The Practice of Kriging," *Advanced Geostatistics in the Mining Industry*, M. Guarascio et al, editors, D. Reidel Publishing Co., Dordrecht-Holland, 1976.

THE VOLUME-VARIANCE RELATIONSHIP: A USEFUL TOOL FOR MINE PLANNING

Dr. Harry Parker, chief geologist and geostatistician,
Fluor Mining & Metals

Knowledge of the amount and average grade of ore in a deposit is vital to the engineer who plans a mining operation, a metallurgical facility, or downstream fabrication plant. Because this knowledge cannot be obtained directly, inferences must be made about the distribution of ore within the deposit from samples obtained by sampling programs. Geostatistical methods provide powerful insights into the prediction of average grade and tonnage.

All mineral deposits are made up of mixtures of ore and waste rock. Various mining methods may be employed to extract ore and waste, with varying degrees of efficiency. If a deposit is mined by hand, the miner can ensure a high degree of segregation of ore and waste material, resulting in a very high-grade, high-unit-cost product at a very low rate of production. On the other hand, if a deposit is mined using large draglines, bucket-wheel excavators, or power shovels, there may be little segregation of ore and waste, and a low-grade, low-unit-cost product will be produced at a high rate of production.

Obviously, an optimum mining method exists for each deposit—a method that yields a product of required average grade for downstream processing at a reasonable cost and at a suitable rate of production. After a mining engineer has analyzed equipment and labor requirements, he can estimate costs and productivity and, using a geostatistical method known as the volume-variance relationship, he can estimate average grade and tonnage that a given mining method will produce. Knowing the average grade and tonnage, he can use discounted cash flow analysis to select a mining method and production rate that maximize the net present worth or return on investment. The volume-variance relationship is also useful for mining and metallurgical planners confronted with the problem of assessing the periodic variability of ore grades in a mining scheme.

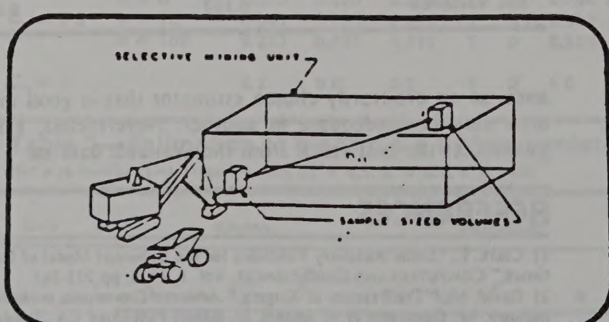


Fig. 1—A selective mining unit is composed of the aggregation of sample-sized volumes, two of which are shown here. The vector h_{ij} separates them.

VOLUME-VARIANCE RELATIONSHIP

Common to all mining methods is the notion of the selective mining unit—the smallest practical volume that can be classified as ore or waste. For a manual mining operation, this unit is obviously a muck-car load, while for a mechanized open-pit operation it might be an individual truck load or even an entire shift's production from a face. To estimate ore reserves, an engineer must know the frequency distribution of the selective mining units.

For most mines, these data are not directly available. Samples are comprised of much smaller volumes, so the problem is to predict the frequency distribution of selective mining units by examining the distribution of appropriate samples.

This is done by assuming that each selective mining unit is comprised of the aggregation of many sample-sized volumes (Fig. 1). Implicitly, the selective mining units will have the same average grade as the sample-sized volumes, without noting at this stage whether they are ore or waste. Schurtz[1] recognized that the variance of the frequency distribution—a parameter that measures the tendency of selective mining units to differ from the mean—will be large if the selective mining units are small in volume. Conversely, the variance will be small if the selective mining units are large in volume. Indeed, the variance vanishes if the selective mining unit is as large as the whole deposit.

The inverse relationship between volume and variance can be seen intuitively by observing that small selective mining units tend to be uniform in grade throughout—some units are all high grade and others all low grade. Thus, strong deviations from the mean may occur. For larger selective mining units, there tends to be a mixture of high and low grade material, and average grades tend to be nearer the average for the whole deposit.

VARIANCE OF THE SMU

The variance of selective mining units (smu) is found using a simple formula:

$$\sigma^2_{smu} = \bar{\gamma}_D - \bar{\gamma}_{smu} \quad (1)$$

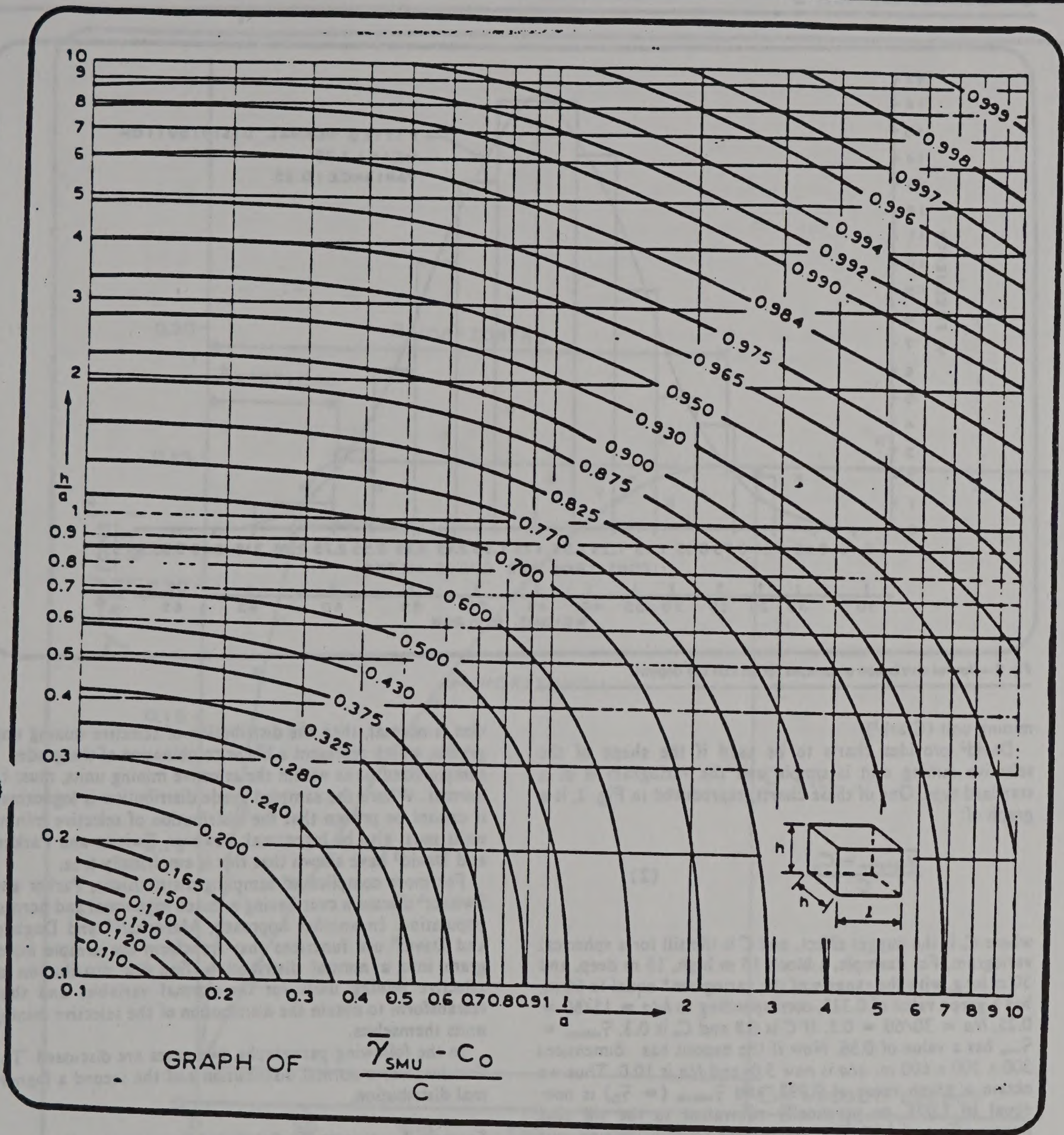


Fig. 2—Graph of $(\bar{\gamma}_{smu} - C_0)/C$ for parallelepiped blocks (David[2]). C is the sill value of the variogram, C_0 is the nugget effect, and a is the range. The dimensions of the block are h and l.

$\bar{\gamma}_D$ is the average value of the variogram function within the deposit and $\bar{\gamma}_{smu}$ is the average value of the variogram function within the selective mining unit. To compute these values, a vector must be drawn between all possible pairs of points lying within the deposit or the selective mining unit, respectively. For each pair, the variogram value is noted, and the average of all of these is computed. In these computations, the vector between points A and B is not assumed to be the same as that between points B and A, even though the variogram values will be the same. Vectors of zero length, such as $A \rightarrow A$ and $B \rightarrow B$, also must be included and assigned a variogram value equivalent to the nugget effect.

Computation of the average variogram values is tedious if done by hand. Frequently the estimator assumes that $\bar{\gamma}_D$ is equivalent to the variance of sample grades (or variogram sill and nugget effect), provided the range of the variogram is short compared with the dimensions of the deposit. To obtain $\bar{\gamma}_{smu}$, simple computer programs can be written that do the calculations for a representative number of points, usually 50, either on a grid or scattered at random within a unit. David[2] discusses this process in more detail and provides a computer program for the purpose. If the grid approach is used, the grid should be defined so that the outermost points are half a grid spacing from the boundary of the selective

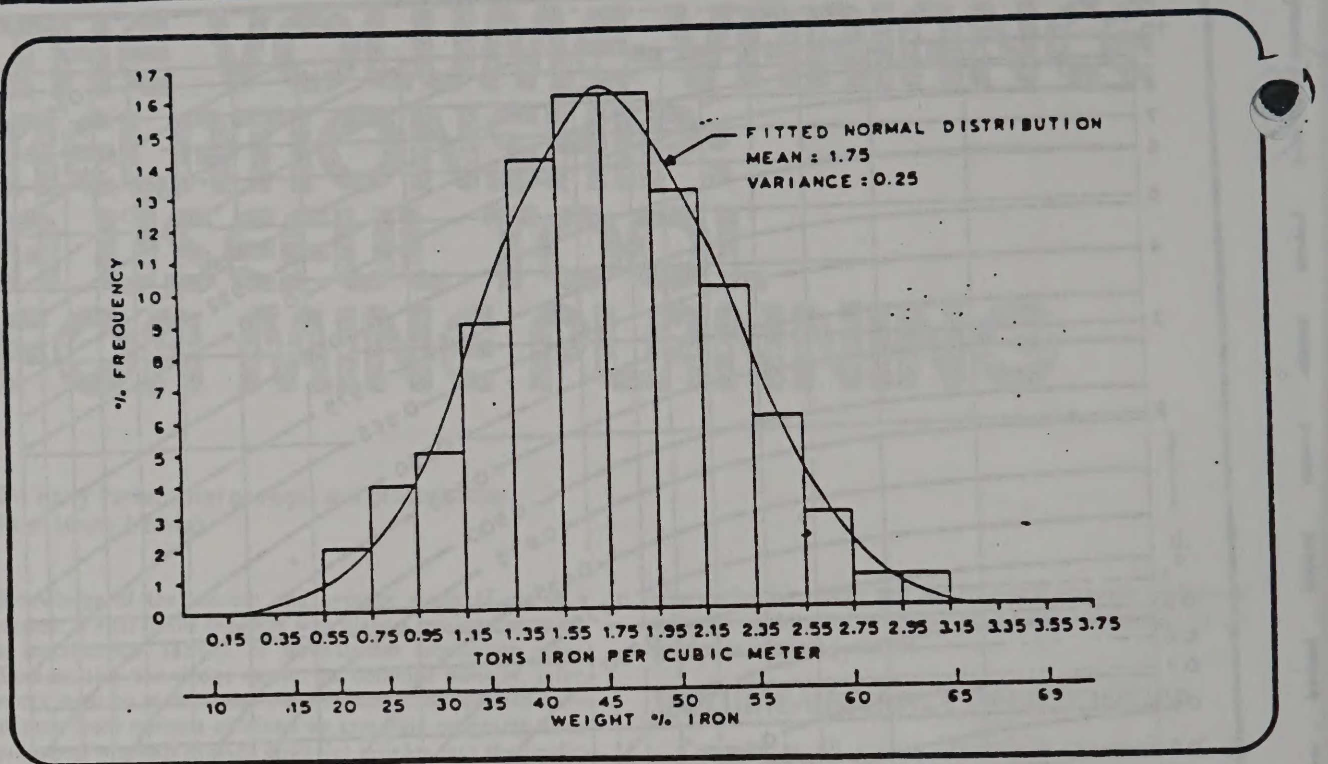


Fig. 3—Normal distribution of samples for an iron ore deposit.

mining unit (Clark[3]).

David[2] provides charts to be used if the shape of the selective mining unit is simple and the variogram is of a standard type. One of these charts, reproduced in Fig. 2, is a graph of:

$$\frac{\bar{\gamma}_{volume} - C_o}{C} \qquad (2)$$

where C_o is the nugget effect, and C is the sill for a spherical variogram. For example, a block 15 m high, 15 m deep, and 30 m long, with the range a of the variogram* equal to 60 m, has a graph value of 0.325, corresponding to $h/a = 15/60 = 0.25$, $l/a = 30/60 = 0.5$. If C is 0.8 and C_o is 0.3, $\bar{\gamma}_{volume} = \bar{\gamma}_{smu}$ has a value of 0.56. Now if the deposit has dimensions 300 x 300 x 600 m, n/a is now 5.0, and l/a is 10.0. Thus we obtain a graph value of 0.998, and $\bar{\gamma}_{volume}$ ($= \bar{\gamma}_D$) is now equal to 1.098, or practically equivalent to the sill plus nugget effect of 1.10. The variance of selective mining unit grades is then $\bar{\gamma} - \bar{\gamma}_{smu} = 1.10 - 0.56 = 0.54$.

SMU FREQUENCY DISTRIBUTION

Estimation of the frequency distribution of selective mining units requires statistical inference. For many mineral deposits, the sample grade distribution approximates a normal or lognormal distribution. If the sampling distribution is normal, then the distribution of selective mining unit grades, which represent a linear combination of the grades sample-sized units within the selective mining units, must be normal. Where the sampled grade distribution is lognormal, it cannot be proven that the distribution of selective mining units must also be lognormal; however, Switzer and Parker[5] and David[2] have shown that this is empirically true.

For more complicated sampling distributions, Parker and Switzer[6] discuss a case having a mixed lognormal and normal population. In another approach, Marechal[7,8] and Dagbert and David[9] use functions that transform the sample histogram into a normal distribution, find the distribution of selective mining units for the normal variable, and then retransform to obtain the distribution of the selective mining units themselves.

In the following paragraphs, two cases are discussed. The first involves a normal distribution and the second a lognormal distribution.

Example 1—a normally distributed iron ore deposit

Fig. 3 illustrates the normal distribution of samples taken from an iron ore deposit, the variograms for which are shown in Fig. 4. Since the density of iron ore varies with grade, any averaging must include weighting by density, which is difficult since density is a nonlinear function of grade. Therefore the units that have been used are density-free tons of iron per cubic meter. The following formula adapted from Parker[10] has been used for conversion:

$$\frac{\text{Tons Fe}}{\text{m}^3} = \frac{A\rho_o\rho_g Z}{100\,(A\rho_o - Z\,(\rho_o - \rho_g))} \qquad (3)$$

where A = percentage of iron in ore mineral, in this case 69.9% Fe in hematite; ρ_o = density of ore mineral, in this case 5.26 gm/cc; ρ_g = density of gangue minerals, in this

*David's chart assumes that the variogram is comprised of dimensionless "point" samples. This assumption is generally acceptable if sample dimensions are less than 1/3 of the corresponding dimension of the block. If sample dimensions are greater, the sample variogram must be transformed to that of a point sample using a process known as regularization (David[2] and Clark[4]). Alternatively, one may use other charts. For instance, if the sample length is equivalent to the height of the block, but the sample dimensions are small enough to be considered as a point in the horizontal dimension, $\bar{\gamma}_{smu}$ may be found from another graph in David[2]. In complex cases, the computer method indicated above is preferred.

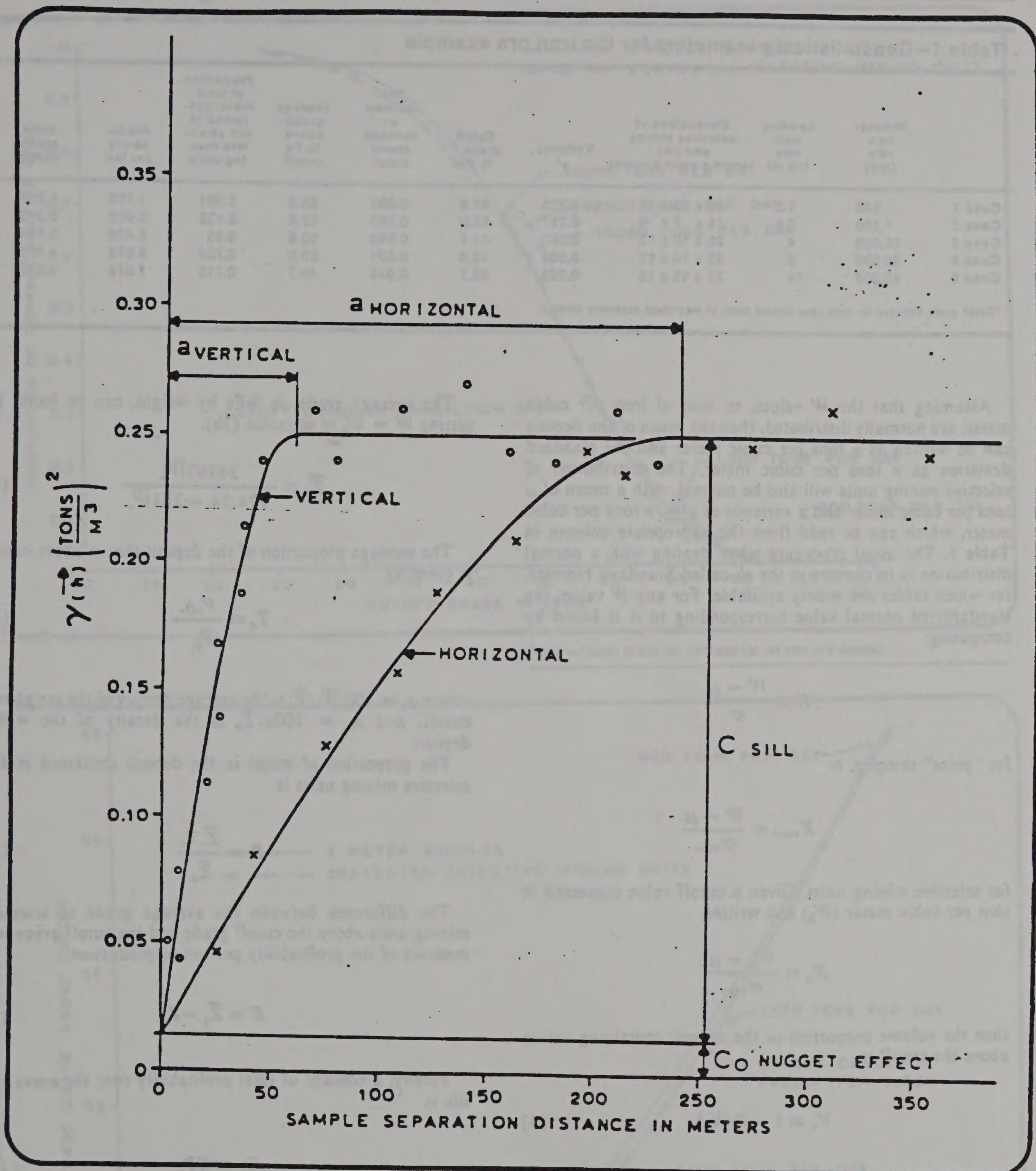


Fig. 4—Variogram for tons of iron per cubic meter.

case 2.65 gm/cc; and Z = weight percent iron.

Using W to represent tons per cubic meter in this example:

$$W = \frac{974.34Z}{36767 - 261Z} \qquad (3a)$$

and, conversely:

$$Z = \frac{36767W}{974.34 + 261W} \qquad (3b)$$

Table I shows the variances and cutoff grades for selective mining units that might typically be associated with operations of various sizes. The variances were derived from a computer program employing the spherical models fitted to the variograms in Fig. 4.

Table 1—Geostatistical parameters for the iron ore example

	Production rate (tpd)	Loading unit size (cu m)	Dimensions of selective mining unit (m) length x width x height	Variance, σ^2	Cutoff grade,* % Fe	Proportion of tonnage above cutoff	Average grade above % Fe cutoff	Proportion of total metal contained in ore selective mining units	Profitability per ton	Total profitability
Case 1	500	1.5	10 x 5 x 6	0.225	67.6	0.001	68.8	0.001	1.195	0.001
Case 2	1,500	2.5	15 x 7 x 9	0.217	55.0	0.103	57.9	0.132	2.952	0.303
Case 3	10,000	4	20 x 10 x 12	0.210	44.1	0.580	50.6	0.65	6.470	3.754
Case 4	20,000	9	25 x 10 x 12	0.209	43.0	0.631	50.0	0.700	6.978	4.403
Case 5	40,000	14	25 x 15 x 15	0.203	42.7	0.648	49.7	0.715	7.076	4.587

*Cutoff grade selected for each case reflects order of magnitude economic studies.

Assuming that the W values, or tons of iron per cubic meter, are normally distributed, then the mean of the deposit can be written as μ tons per cubic meter and the standard deviation as σ tons per cubic meter. The distribution of selective mining units will also be normal, with a mean of μ tons per cubic meter and a variance of σ^2_{smu}/n tons per cubic meter, which can be read from the appropriate column in Table 1. The usual procedure when dealing with a normal distribution is to convert to the so-called Standard Normal, for which tables are widely available. For any W value, the standardized normal value corresponding to it is found by computing:

$$X = \frac{W - \mu}{\sigma}$$

for "point" samples, or

$$X_{smu} = \frac{W - \mu}{\sigma_{smu}}$$

for selective mining units. Given a cutoff value expressed in tons per cubic meter (W_c) and writing

$$X_c = \frac{W_c - \mu}{\sigma_{smu}}$$

then the volume proportion of the deposit containing values above the cutoff is

$$V_c = 1 - \Phi(X_c) \qquad (4)$$

where $\Phi(\ )$ is the cumulative probability function for a standard normal variable, which is available in most statistics tables and text books.

The average value (tons per cubic meter) of the proportion of the deposit above cutoff is

$$\overline{W}_c = \mu + \frac{\sigma_{smu}}{V_c} f(X_c) \qquad (5)$$

where $f(X_c)$ is the height of the normal probability density curve at X_c and may be calculated by

$$f(X_c) = \frac{1}{2\pi} \exp\left(-\frac{1}{2} X_c^2\right)$$

The average grade, in %Fe by weight, can be found by setting $W = \overline{W}_c$ in equation (3b).

$$\overline{Z}_c = \frac{36767\overline{W}_c}{974.34 + 261\overline{W}_c} \qquad (6)$$

The tonnage proportion of the deposit that is above cutoff is given by

$$T_c = \frac{V_c \rho_c}{\rho_\mu} \qquad (7)$$

where $\rho_c = 100\,\overline{W}_c/\overline{Z}_c$ is the average density of the ore above cutoff, and $\rho_\mu = 100\mu/\overline{Z}_o$ is the density of the whole deposit.

The proportion of metal in the deposit contained in the selective mining units is

$$R = \frac{\overline{Z}_c T_c}{\overline{Z}_o} \qquad (8)$$

The difference between the average grade of selective mining units above the cutoff grade and the cutoff grade is a measure of the profitability per ton of production:

$$E = \overline{Z}_c - Z_c \qquad (9)$$

Finally, a concept of total profitability over the operating life is

$$E_T = ET_c \qquad (10)$$

These variables are graphed as a function of cutoff grade in Figs. 5-9. It is important to observe that the curve based on samples is nearly identical to that based on selective mining units. From Table 1, it is clear that the variance of selective mining units does not vary markedly with their size, hence, only the curve for case 3 (10,000 tpd) has been plotted on the graphs. The reason for this relative lack of variance is that many sedimentary iron ore deposits show strong continuity of grade on the scale of selective mining units, meaning that the range of the variogram is extremely large with respect to the block size. The averaging effect that should reduce the variance is therefore weak.

Curves are also shown that relate cutoff grade to produc-

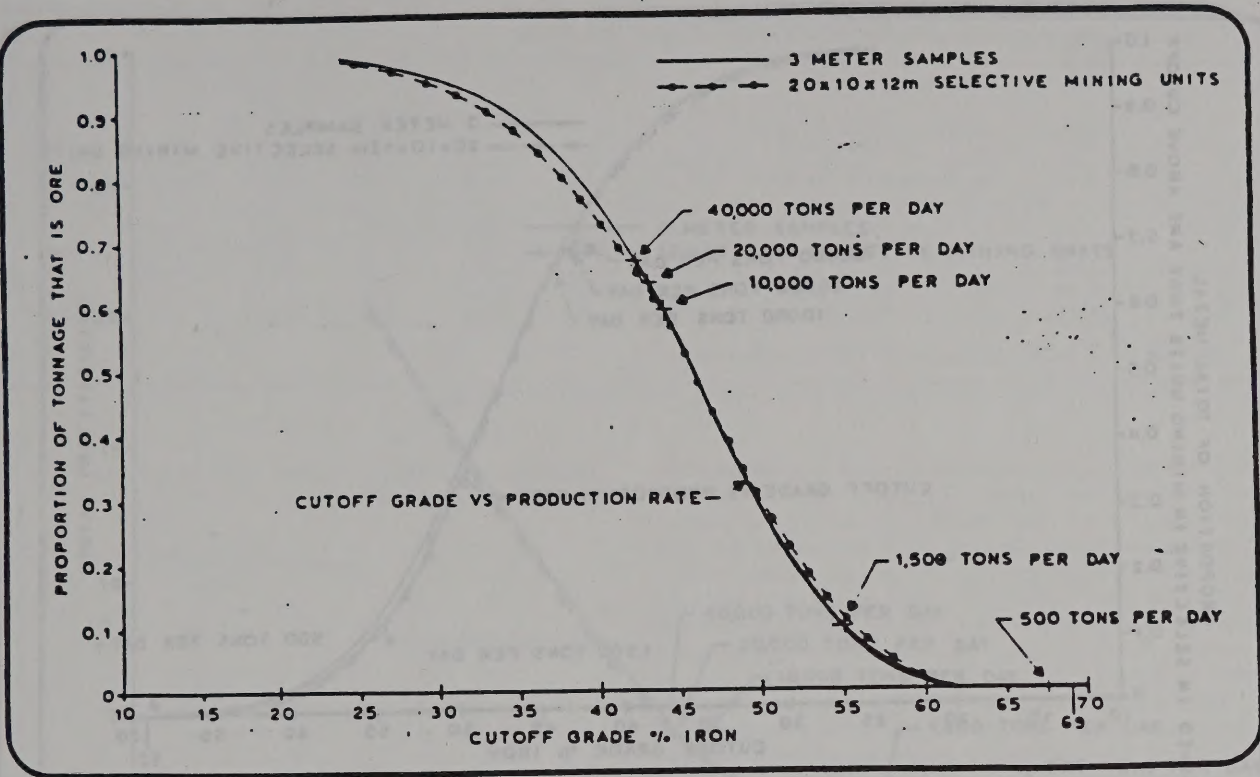


Fig. 5—Cutoff grade vs. tonnage for an iron ore deposit.

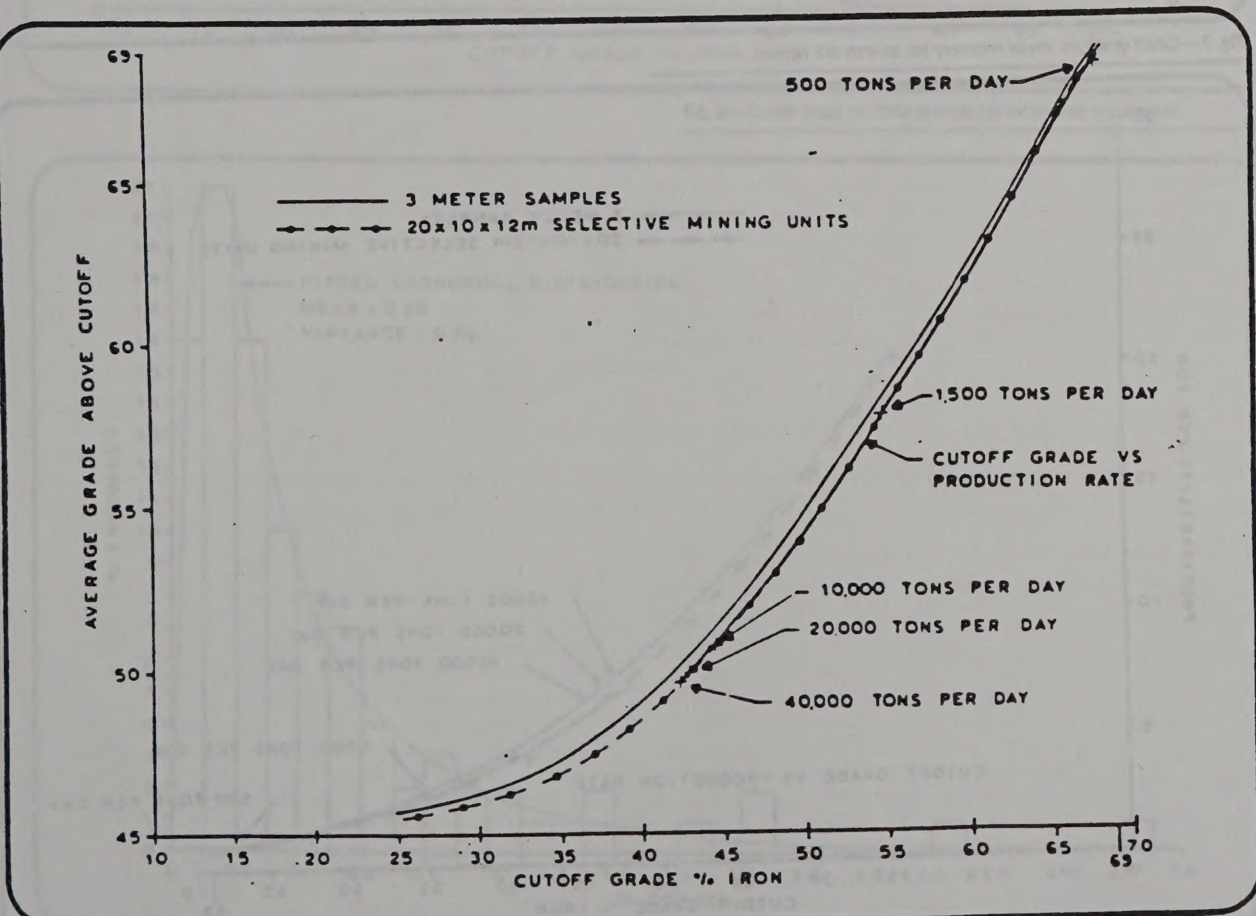


Fig. 6—Cutoff grade vs. average grade for an iron ore deposit.

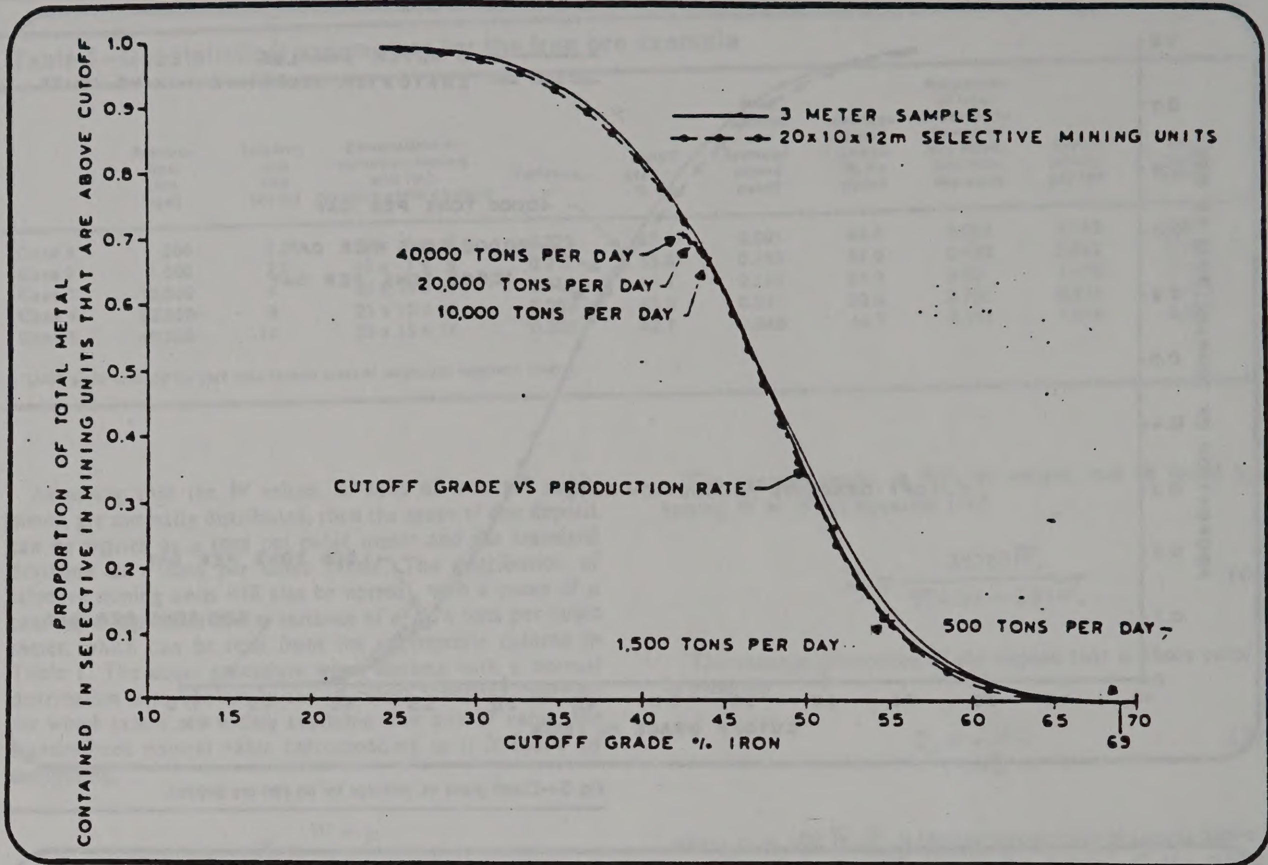


Fig. 7—Cutoff grade vs. metal recovery for an iron ore deposit.

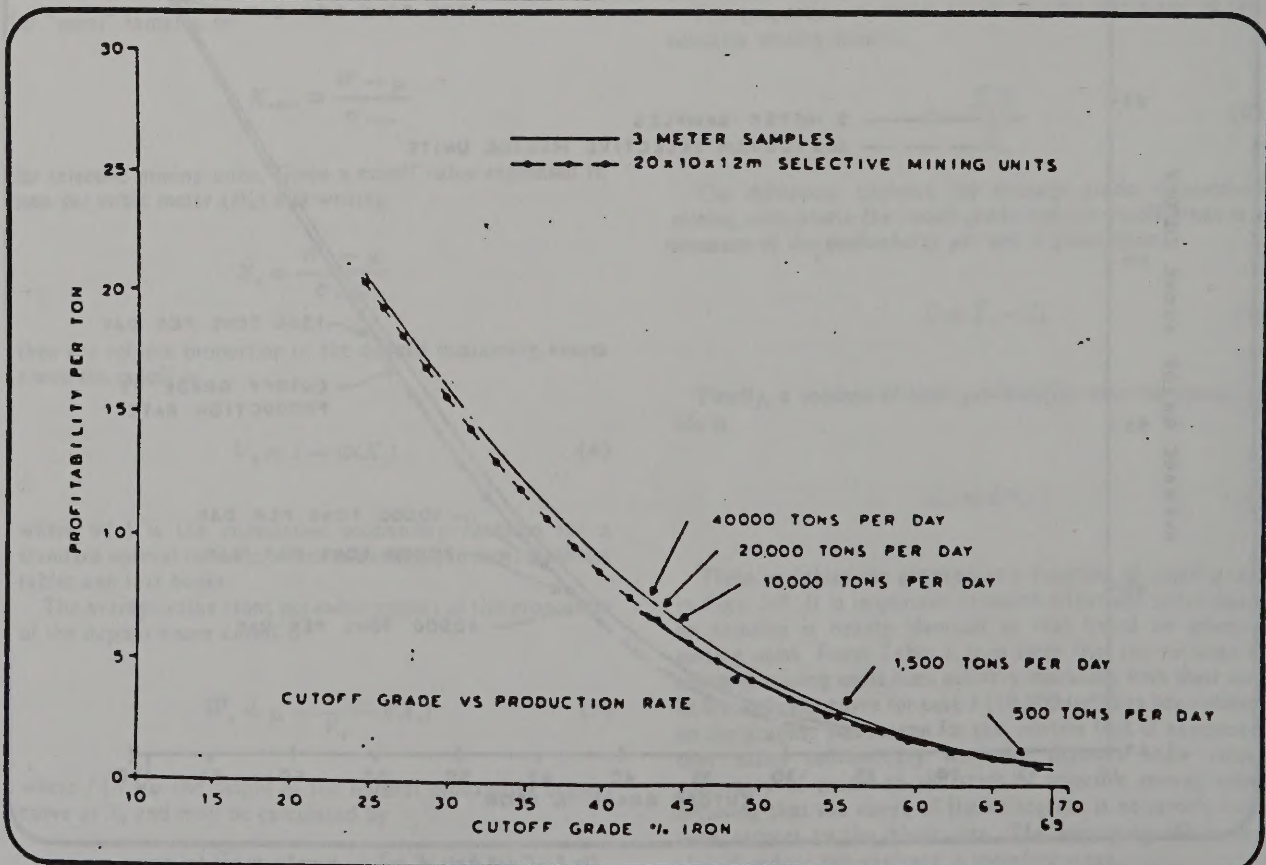


Fig. 8—Cutoff grade vs. profitability per ton for an iron ore deposit.

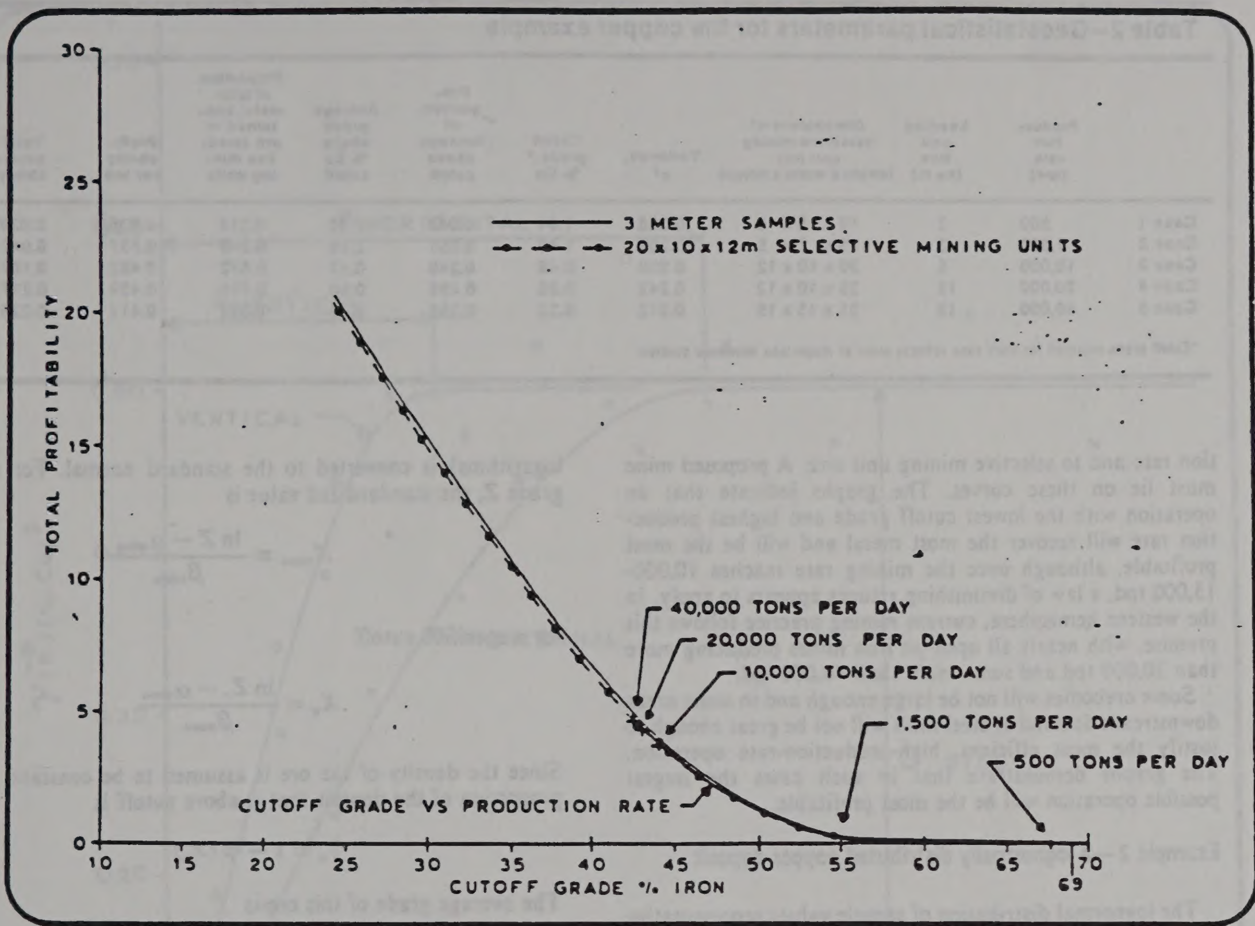


Fig. 9—Cutoff grade vs. total profitability for an iron ore deposit.

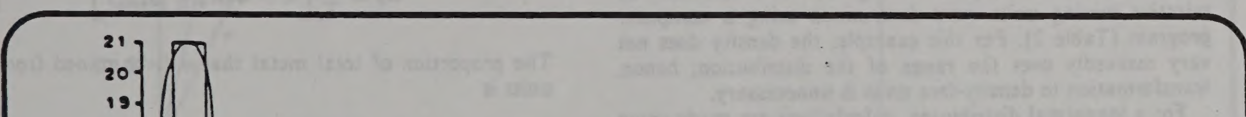


Fig. 10—Lognormal appearing distribution of samples for a copper deposit.

Table 2—Geostatistical parameters for the copper example

	Production rate (tpd)	Loading unit size (cu m)	Dimensions of selective mining unit (m) length x width x height	Variance, σ^2	Cutoff grade,* % Cu	Proportion of tonnage above cutoff	Average grade above % Cu cutoff	Proportion of total metal contained in ore selective mining units	Profitability per ton	Total profitability
Case 1	500	2	10 x 5 x 6	0.356	1.64	0.042	2.58	0.214	0.936	0.039
Case 2	1,500	3	15 x 7 x 9	0.305	1.40	0.055	2.19	0.240	0.787	0.043
Case 3	10,000	5	20 x 10 x 12	0.259	0.49	0.346	0.97	0.672	0.482	0.167
Case 4	20,000	15	25 x 10 x 12	0.242	0.36	0.495	0.80	0.791	0.439	0.217
Case 5	40,000	19	25 x 15 x 15	0.212	0.33	0.555	0.75	0.822	0.411	0.228

*Cutoff grade selected for each case reflects order of magnitude economic studies.

tion rate and to selective mining unit size. A proposed mine must lie on these curves. The graphs indicate that an operation with the lowest cutoff grade and highest production rate will recover the most metal and will be the most profitable, although once the mining rate reaches 10,000-15,000 tpd, a law of diminishing returns appears to apply. In the western hemisphere, current mining practice follows this premise, with nearly all open-pit iron mines producing more than 20,000 tpd and some more than 40,000 tpd.

Some orebodies will not be large enough and in some areas downstream demand at steel mills will not be great enough to justify the most efficient, high-production-rate operation. The graphs demonstrate that in such cases the largest possible operation will be the most profitable.

Example 2—a lognormally distributed copper deposit

The lognormal distribution of sample values representative of a disseminated copper deposit are shown in Fig. 10 and the variograms for the deposit in Fig. 11. The variances of selective mining units were determined using a computer program (Table 2). For this example, the density does not vary markedly over the range of the distribution; hence, transformation to density-free units is unnecessary.

For a lognormal distribution, calculations are made using the mean and standard deviation of the logarithms of the grades, not the grades themselves. Writing μ and σ for the mean and standard deviation of the sample grades, then

$$\alpha = \ln \mu - \frac{1}{2}\beta^2$$

$$\beta^2 = \ln\left(\frac{\sigma^2}{\mu^2} + 1\right)$$

where α and β are the mean and standard deviation of the logarithms of the grades. Assuming that the selective mining units are also lognormally distributed, the mean μ will remain constant, but the standard deviation will become σ_{smu}. Therefore both α and β will change with block size.

$$\alpha_{smu} = \ln \mu - \frac{1}{2}\beta^2_{smu}$$

$$\beta^2_{smu} = \ln\left(\frac{\sigma^2_{smu}}{\mu^2} + 1\right) \qquad (11)$$

Sichel[11] and Link, Koch, and Schuenemeyer[12] provide tables for estimating these parameters if the number of samples is small.

As in the previous example, the distribution (now of the logarithms) is converted to the standard normal. For any grade Z, the standardized value is

$$X_{smu} = \frac{\ln Z - \alpha_{smu}}{\beta_{smu}}$$

for a specified cutoff

$$X_c = \frac{\ln Z_c - \alpha_{smu}}{\beta_{smu}}$$

Since the density of the ore is assumed to be constant, the proportion of the deposit that is above cutoff is

$$T_c = 1 - \Phi(X_c) \qquad (12)$$

The average grade of this ore is

$$\overline{Z}_c = \frac{\mu}{T_c}\left[1 - \Phi(X_c - \beta_{smu})\right] \qquad (13)$$

The proportion of total metal that will be mined from ore units is

$$R = \frac{\overline{Z}_c T_c}{\mu} \qquad (14)$$

The profitability factor for mining is

$$E = \overline{Z}_c - Z_c \qquad (15)$$

Finally, the measure of total profitability is, as before

$$E_T = ET_c \qquad (16)$$

In Figs. 12-16, these variables are graphed as a function of cutoff grade for the cases listed in Table 2. Overlain is a curve for cutoff grade versus size of selective mining unit typically associated with a given production rate.

In this example, the variance for selective mining units is significantly lower than the variance for samples, and predictions made on the curve for samples may be seriously in error. For instance, if the cutoff grade is 0.4% Cu, the samples predict 38% of the deposit will be ore, and the average grade will be 1.04% copper. However, at a 40,000-tpd production rate, the ore tonnage will be 46% of the deposit, and the average grade will be only 0.82% copper.

The graphs indicate that while large-scale operations recover the greatest proportion of the total metal in the

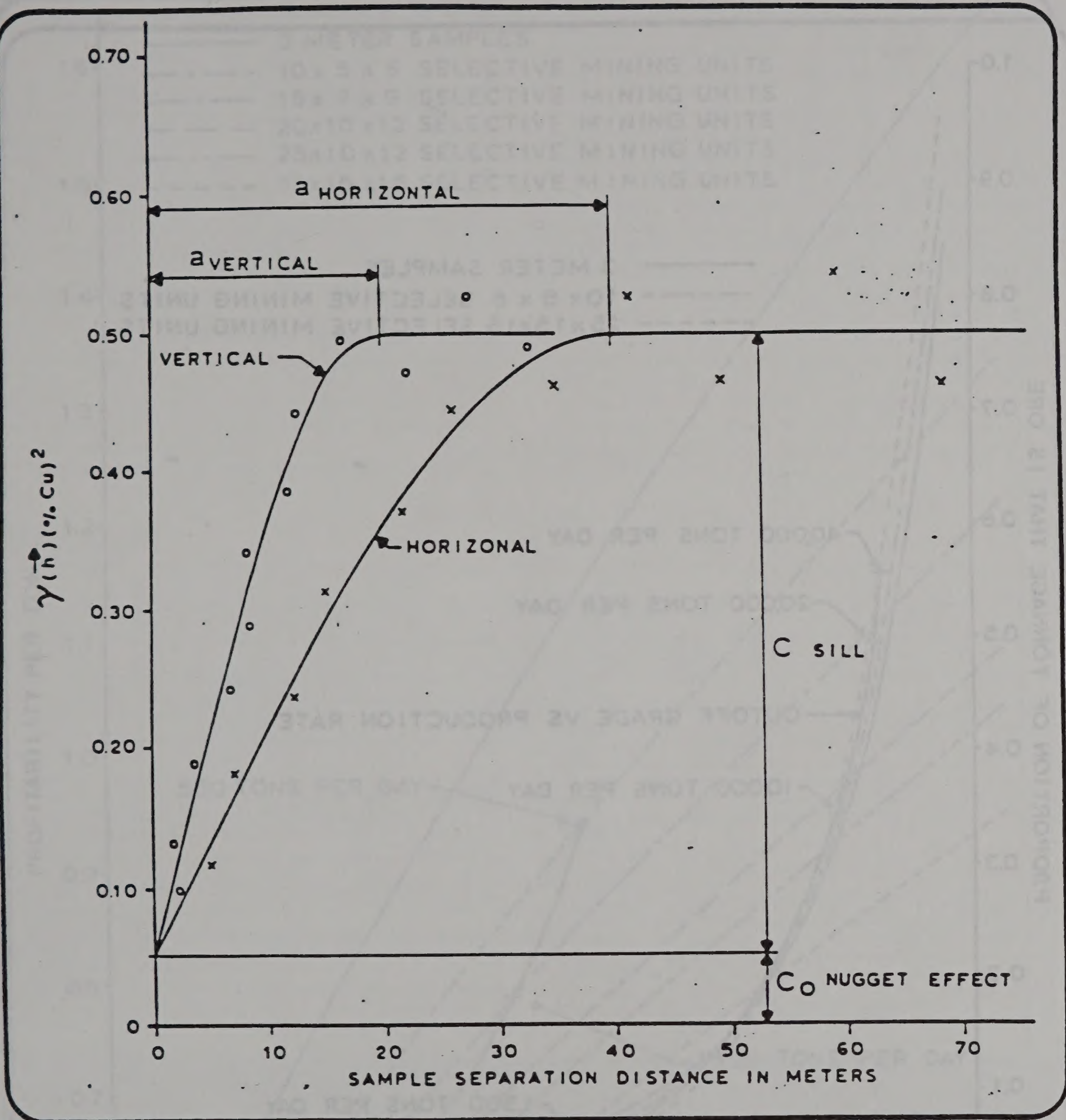


Fig. 11—Variogram for percent copper.

deposits and have the greatest total profitability, profitability per ton is least for high production rates. Clearly, if downstream demand for product is low, or if there are a large number of deposits available to satisfy the demand, high-cutoff-grade, low-production-rate operations will be most profitable in the short term. In the longer term, lower-cutoff-grade, higher-production-rate operations will recover more metal and have greater total profitability.

Until recently, most mining companies have tended to follow the latter course. However, during the last five years, the costs of mine and metallurgical facilities have risen to the extent that profitability translated into net revenue is sometimes incapable of repaying them within a reasonable time. As a result, many new projects have scaled down production rates.

More sophisticated analysis requires construction of multidimensional economic models, employing grade-tonnage curves for each production rate. Then, a calculation of net present worth or return on investment as a function of grade, tonnage, production rate, metal price, and percent-loan financing can be made. The values of these parameters that maximize net present worth or return on investment can then be determined and project development planned accordingly. A recent case study was presented by Recny.[13]

The previous discussion has assumed that the mineralized area, here called the "deposit," does not change shape from case to case. Examples would be deposits where ore is roughly equally accessible to extraction, regardless of cutoff grade, such as near-surface deposits of uranium, bauxite, or nickel laterite and many underground deposits. For deposits

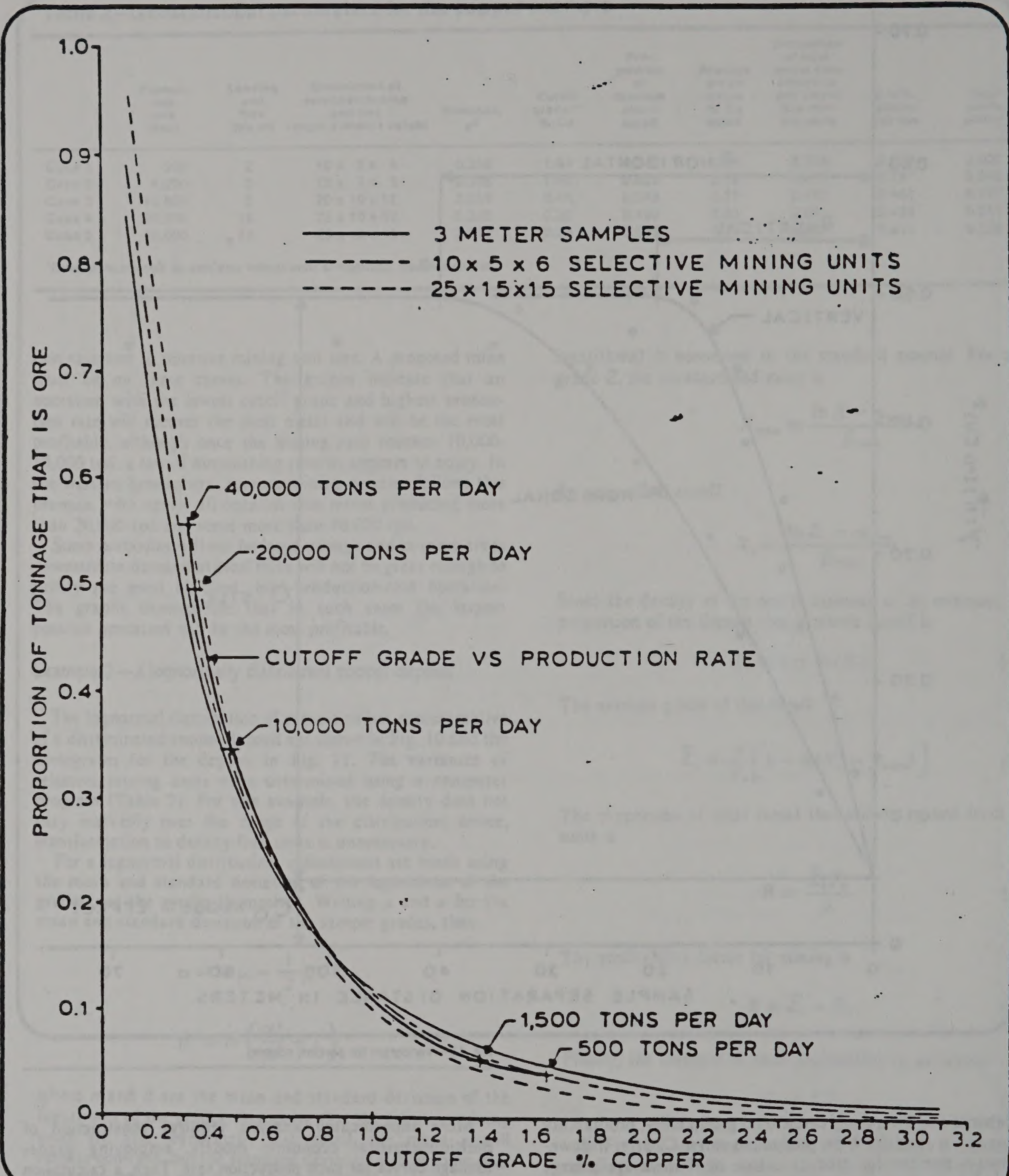


Fig. 12—Cutoff grade vs. tonnage for a copper deposit.

with zoned mineralization and/or deep overburden, such as porphyry coppers, the frequency distribution and variogram must be calculated for each case to reflect only those samples lying within a proposed mining area.

EXTENDING THE V-V RELATIONSHIP

In production planning, it is often necessary to predict the variability of selective mining units within an area of a deposit that is considerably smaller than the deposit as a whole. For instance, we may wish to know the variability of daily production units within a month's time. The variance in mill feed grades over a month's time is

$$\sigma^2_{smu/V} = \bar{\gamma}_{\substack{\text{volume}\\ \text{of}\\ \text{month's}\\ \text{production}}} - \bar{\gamma}_{smu} \qquad (17)$$

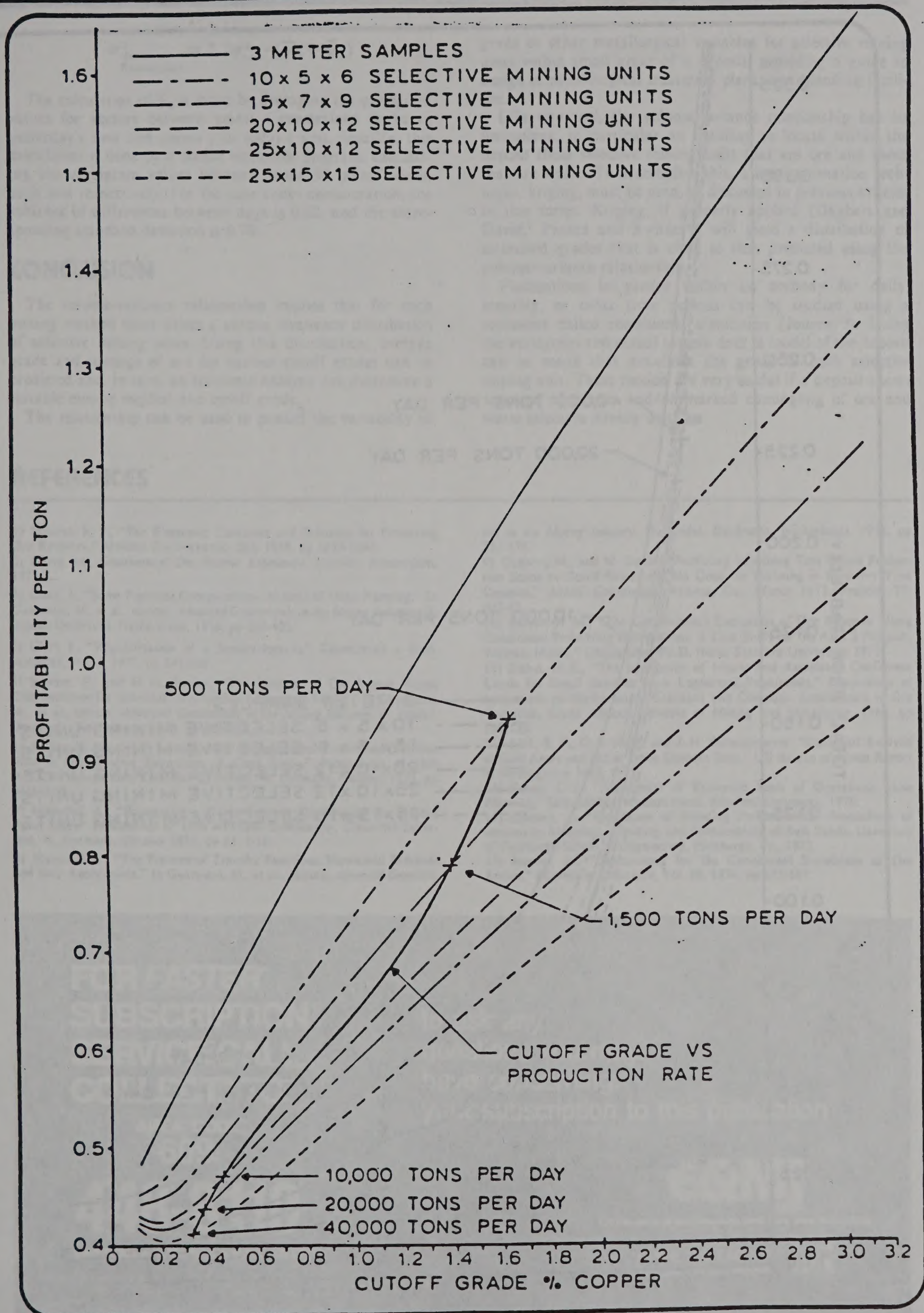


Fig. 15—Cutoff grade vs. profitability per ton for a copper deposit

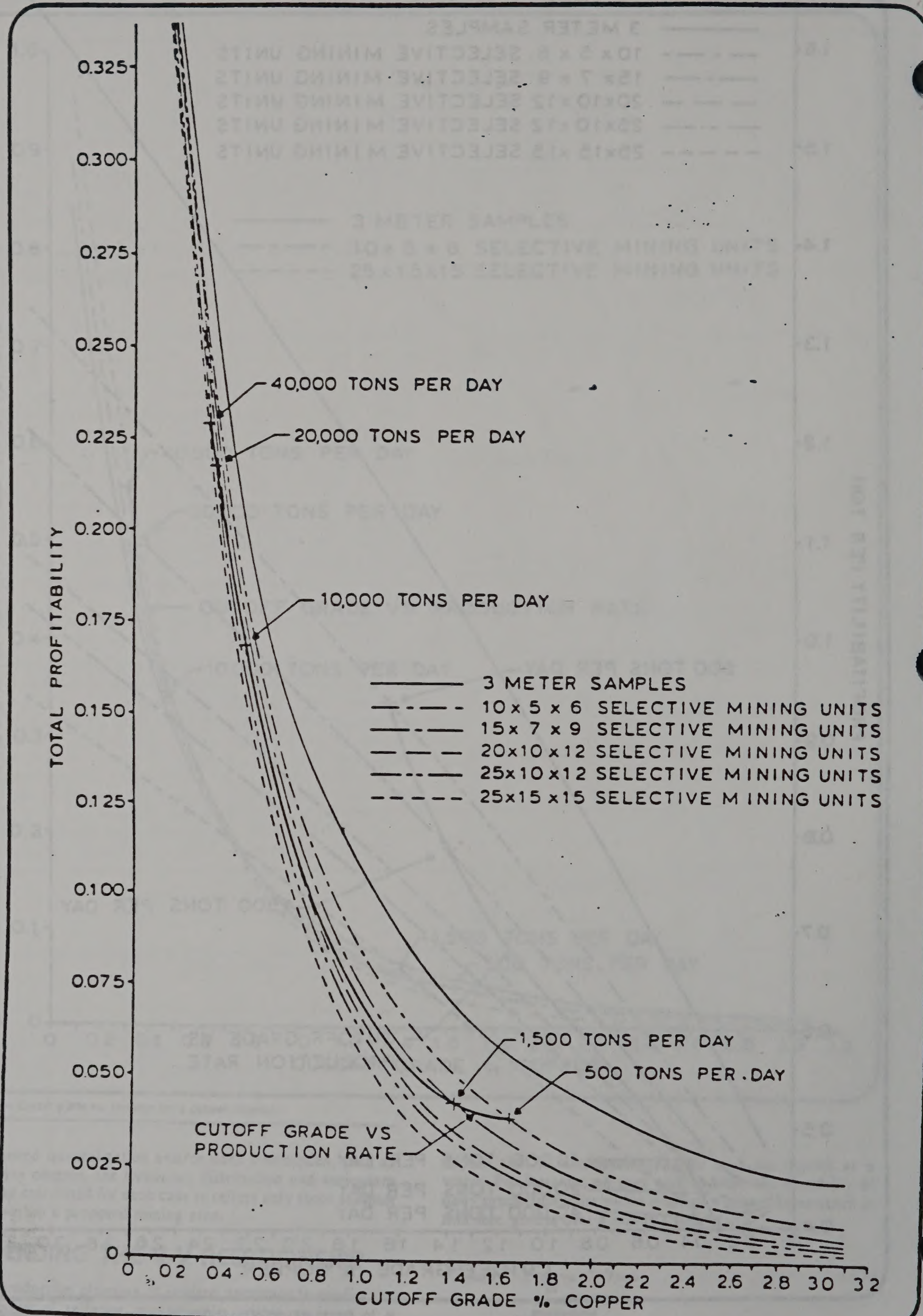


Fig. 16—Cutoff grade vs. total profitability for a copper deposit.

$$\sigma^2_{\substack{\text{daily}\\ \text{fluctuation}}} = 2\ (\sigma^2_{\text{smu}} - \bar{\gamma}_D + \bar{\gamma}_{ij})$$

The calculation of $\bar{\gamma}_{ij}$ is made by averaging the variogram values for vectors between points i constrained to be in yesterday's unit and points j in today's unit. Usually, this calculation is done by a simple computer program, calculating the variogram values between about 50 points within each unit respectively. For the case under consideration, the variance of differences between days is 0.62, and the corresponding standard deviation is 0.78.

CONCLUSION

The volume-variance relationship implies that for each mining method there exists a unique frequency distribution of selective mining units. Using this distribution, average grade and tonnage of ore for various cutoff grades can be predicted and, in turn, an economic analysis can determine a suitable mining method and cutoff grade.

The relationship can be used to predict the variability of grade or other metallurgical variables for selective mining units within small areas of a deposit, providing a guide to design of metallurgical treatment plants and blending facilities.

Like any tool, the volume-variance relationship has its limitations, in particular an inability to locate within the deposit those selective mining units that are ore and those that are waste. To accomplish this, a local estimation technique, kriging, must be used, as discussed in previous articles in this series. Kriging, if properly applied (Dagbert and David,[9] Parker and Switzer[6]), will yield a distribution of estimated grades that is close to that predicted using the volume-variance relationship.

Fluctuations in grades within an orebody for daily, monthly, or other time periods can be studied using a technique called conditional simulation (Journel[15]). Using the variogram and actual sample data, a model of the deposit can be made that simulates the grade of each selective mining unit. These models are very useful if a deposit shows zonation of grades and/or marked comingling of ore and waste selective mining units.

REFERENCES

1) Schurtz, R. F., "The Electronic Computer and Statistics for Predicting Ore Recovery," MINING ENGINEERING, Oct. 1959, pp 1035-1044.

2) David, M., *Geostatistical Ore Reserve Estimation*, Elsevier, Amsterdam, 1977.

3) Clark, I., "Some Practical Computational Aspects of Mine Planning," In Guarascio, M., et al., editors, *Advanced Geostatistics in the Mining Industry*, D. Reidel, Dordrecht, Netherlands, 1976, pp 391-402.

4) Clark I., "Regularization of a Semivariogram," COMPUTERS & GEOSCIENCES, Vol. 3, 1977, pp 341-346.

5) Switzer, P., and H. M. Parker, "The Problem of Ore Versus Waste Discrimination for Individual Blocks: The Lognormal Model," In Guarascio, M., et al., editors, *Advanced Geostatistics in the Mining Industry*, D. Reidel, Dordrecht, Netherlands, 1976, pp 203-218.

6) Parker, H. M., and P. Switzer, "Use of Conditional Probability Distributions in Ore Reserve Estimation, A Case Study," Proceedings of 13th APCOM Symposium, Clausthal-Zellerfeld, W. Germany, October 1975, pp MII: 1-16.

7) Marechal, A., "Forecasting a Grade-Tonnage Distribution for Various Panel Sizes," Proceedings of 13th APCOM Symposium, Clausthal-Zellerfeld, W. Germany, October 1975, pp EI: 1-18.

8) Marechal, A., "The Practice of Transfer Functions: Numerical Methods and their Applications," In Guarascio, M., et al., editors, *Advanced Geostatistics in the Mining Industry*, D. Reidel, Dordrecht, Netherlands, 1976, pp 253-276.

9) Dagbert, M., and M. David, "Predicting Vanishing Tons Before Production Starts or Small Blocks are No Good for Planning in Porphyry Type Deposits," AIME Convention, Atlanta, Ga., March 1977, Preprint 77-AO-83.

10) Parker, H. M., "The Geostatistical Evaluation of Ore Reserves Using Conditional Probability Distributions: A Case Study for the Area 5 Prospect, Warren, Maine," Unpublished Ph.D. thesis, Stanford University, 1975.

11) Sichel, H. S., "The Estimation of Means and Associated Confidence Limits for Small Samples from Lognormal Populations," Proceedings of symposium on Mathematical Statistics and Computer Applications in Ore Valuation, South African Institute of Mining and Metallurgy, 1966, pp 106-123.

12) Link, R. F., G. S. Koch, and J. H. Schuenemeyer, "Statistical Analysis of Gold Assay and Other Trace Element Data," US Bureau of Mines Report of Investigation 7495, 1971.

13) Recny, C. J., "Assessment of Economic Scale of Operations Mine Planning," Unpublished Masters thesis, Stanford University, 1978.

14) Elbrond, J., "Evaluation of Blending Performance," Proceedings of seminar on Sampling, Blending, and Proportioning of Bulk Solids, University of Pittsburgh School of Engineering, Pittsburgh, Pa., 1973.

15) Journel, A., "Geostatistics for the Conditional Simulation of Ore Bodies," ECONOMIC GEOLOGY, Vol. 69, 1974, pp 673-687.

GEOSTATISTICAL SIMULATION: METHODS FOR EXPLORATION AND MINE PLANNING

A. G. Journel, associate professor, Stanford University

A serious challenge facing the mineral industry in the next decade will be ore production from new marginal deposits. Lower average grades require greater rates of production, more sophisticated milling techniques, and larger investments. Investors will therefore require greater accuracy in the prediction of total tonnage and grades from sample data. Simultaneously, a reduction in the maximum error limits designed into the mining and processing facilities will be necessary for the marginal deposits to be put into production.

The recently developed technique of "Conditional Simulations" provides numerical models of any deposit. Just as one may use aircraft models in a wind tunnel to simulate flight performance, deposit models may be used to test the efficiency of any planned survey or mining procedure as well as their robustness with respect to the various unknowns about the real deposit. The mining plan can then be better adapted, or the need for additional information may be specified. Optimization of these simulation methods and development of their full potential in mine planning represent an exciting opportunity for the next decade.

COMPENSATING FOR THE UNKNOWN

During the past decade, the capital investment required to bring new ore deposits into production has risen sharply. In countries with favorable political and fiscal environments, new deposits are characterized by lower average grades and more stringent economic limits for mining and milling. New projects based on these deposits do not have a 10% or 20% margin for error, as was the case in the past. Also, inflation in capital investments requires robust planning, i.e. the calculated economic feasibility should be maximally independent of the uncertainty inherent in unknown factors. The unknowns are numerous: political and social conditions, stability of the market, and quantity and quality of the ore that the future mine will effectively recover.

Unknowns about the ore itself are particularly important—there is a sizeable list of recently started operations that are running into critical problems because recovered grades and tonnages were inaccurately forecasted in the feasibility studies. It is therefore very important to analyze the sensitivity of the principal production figures to the

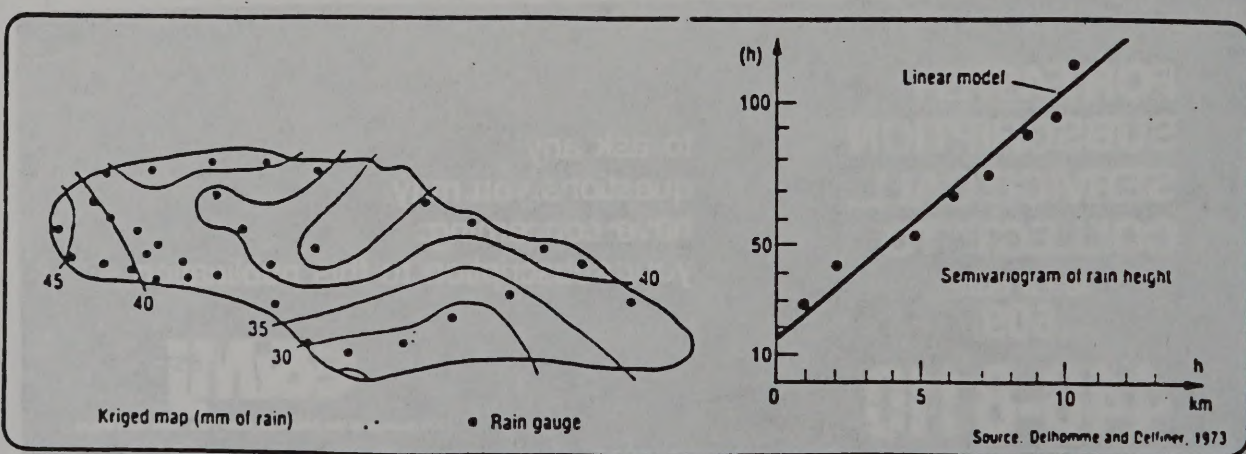
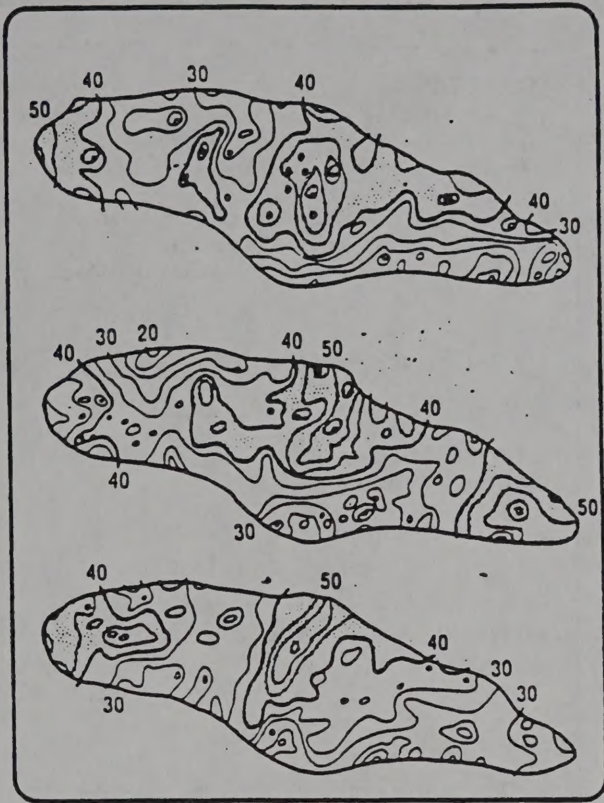


Fig. 1—Estimation $\hat{D}$ of a particular rainstorm D. The precipitation levels corresponding to one rainstorm in the desert basin of Kadjemeur, Chad, have been recorded at various gauge locations. A linear model, fitted to the corresponding experimental semivariogram $\gamma(h)$, characterizes the spatial variability of the phenomenon. The model has been used to estimate (by kriging) the unknown precipitation levels at each node of a dense grid. These estimates are used to draw the contour map of $\hat{D}$. As an example more related to mining, the maps of Figs. 1 and 2 could have been presented as isopach maps. The variable under study would then be the thickness of a seam.

Fig. 2—Three simulated rainstorms D_1, D_2, D_3. The three simulations show the same linear model characteristic of the spatial variability measured on the real precipitation map (Fig. 1). Simulated values of precipitation levels equal the real values recorded at those locations for which data are available. Because of the lack of real data, the three simulations can differ considerably at the right tip of the basin. If more than three rainstorms were simulated and then the data were averaged together, the resulting average map would be identical to the kriged map in Fig. 1. Note that the three maps show a degree of variability over short distances that is not present in the estimated map $\dot{D}$. Kriging can thus be seen to "smooth" the spatial variability of reality D.



unknown parameters before deciding on a project. Such sensitivity analysis with regard to the market of the metal is usually done, but this is not enough. Other major unknowns include tonnages and grades.

Even if the data and the estimation procedures used are good, estimates will depart somewhat from reality. These departures may be characterized through the concept of estimation variance provided by geostatistics. However, estimation variance is merely a number, and it does not express well the consequences of a given departure on the various links of the project. Simulation provides a better picture.

Let's start with a simple example: The average grade of a particular block is estimated to be 1% Cu with a standard deviation of estimation σ = 0.2% Cu. Assuming a normal distribution for the error, the true grade Z of the block is normally distributed around the estimator $\dot{Z}$ = 1%, with a standard deviation σ = 0.2. Simulation of a normal random variable with mean $\dot{Z}$ and variance σ^2 will provide a series of "possible" values for Z, say $Z_1 = 0.7$, $Z_2 = 0.95$, $Z_3 = 1.2$. Any of the previous three (or more) simulated values can be the true value of Z. Hence a plan for mining out that block should consider not only the estimator $\dot{Z}$, which is the mean of all "possible" values, but also the possibility of having either Z_1, Z_2, or Z_3. In other words, the economic viability of the project should be robust with regard to the unknown grade Z.

Another example: An estimated image $\dot{D}$ can be built of the whole deposit D. Usually this image consists of a number of estimated maps, sections, tonnage-grade curves, etc. Simulations of the deposit consist of a series of other "possible" images D_1, D_2, D_3, etc. If these simulated images are all equally probable, then any one of them could match the reality D; thus, the planned mining project should be efficient not only for $\dot{D}$ but also for D_1, D_2, D_3, etc.

It should be noted that the greater the quantity of available information, the smaller will be the discrepancies between the various simulated images; by extension, if the information were perfect (i.e. an infinite amount of accurate data), all images would be identical and equal to the reality: $D_1 = D_2 = D_3 = \cdots = \dot{D} = D$. Hence, a sensitivity analysis will consist of applying the planned project to the various simulated deposits D_1, D_2, D_3, etc., and testing its efficiency and robustness in each case. If the project works well on all simulated deposits, then investments can proceed safely. If not, either some parameters of the project should be modified to allow for a greater robustness, or additional information should be specified in order to reduce the discrepancies between possible simulated images.

MODELING AND SIMULATING

A model, both deterministic and probabilistic, of a deposit can be built based on the detailed geological survey and the geostatistical analyses of available numerical data (grades, thicknesses, densities, etc.). The probabilistic technique of Conditional Simulations, extensively described in Journel (1974) and Journel and Huijbregts (1978), then provides several "possible" realizations or images of that initial model. Each such image represents a simulated deposit D_i similar to the real deposit D. The simulation will exhibit the same geological and geostatistical features, and simulated values (grades, thicknesses, etc.) will equal the real data values at those locations where real data have been obtained (see Fig. 2). These simulated deposits are almost "perfectly known," in this case through knowledge of a very dense data grid (5 x 5 m), whereas the real deposit D may only be surveyed on a widely spaced grid (100 x 100 m) with some information available at small spacing (e.g. through a cross of holes at 5-m spacing).

In a way, constructing a simulation consists of filling in the nodes of a dense grid with simulated data consistent with the spatial continuity of the few real data available. Estimation $\dot{D}$ also consists of such a filling-in operation, but the estimated data do not generally fulfill the consistency requirement: the estimated map $\dot{D}$ in Fig. 1 appears as a smoothed image of reality D, which could be any one of the three simulations D_1, D_2, or D_3 in Fig. 2.

On each of the simulated and almost perfectly known deposits D_i, it is possible to perform a simulation of the planned chain of production, including mining, hauling, blending, mill feeding, and economic analyses. Such simulations allow advance control of the planned project in terms of

CORRECTIONS

The following are corrections for Dr. H. Parker's paper, "The Volume-Variance Relationship," presented as Geostatistics Part 5 in the October 1979 issue of E&MJ:

- Page 110, line 6—The test indicates a variance of σ^2_{smu}/n. The correct term is simply σ^2_{smu}.
- Page 110, bottom left—The equation should read:

$$f(X_c) = \frac{1}{\sqrt{2\pi}} \exp\left(-\frac{1}{2}X_c^2\right)$$

- Page 120, bottom right—The equation should read:

$$\frac{\sigma^2_{smu}/V}{2} \text{ or } 0.44/2$$

- Page 123, top left—The equation should read:

$$\sigma^2_{\substack{\text{daily}\\\text{fluctuation}}} = 2\left(\sigma^2_{\substack{\text{daily}\\\text{production}}} - \overline{\gamma}_D + \overline{\gamma}_{ij}\right)$$

In Dr. I. Clark's paper, "The Semivariogram—Part 1," published in the July 1979 issue of E&MJ, the equation at the bottom left part of p 92 should read:

$$\gamma^*(h) = \frac{1}{2(N-h)} \sum_{i=1}^{N-h} (g_i - g_{i+h})^2$$

both efficiency and robustness with regard to the various "possible" deposits D_i. If this robustness is deemed either insufficient or too costly, then the study's conclusion may be that more information (e.g. more drillholes or more pilot tests on bulk samples) is required to better focus the project. Once this information is obtained, the model of the deposit is refined before new simulations are performed.

RANGE OF APPLICATIONS

Although its development has been quite recent, the technique of Conditional Simulations has had numerous applications ranging from exploration to production planning. Examples include:

1) A study of the number and possible spatial arrangements of geochemical anomalies in a given province, and their survey, cf. Dagbert (1978). The variables simulated are continuous concentrations of major elements in a pluton.

2) Studies of the sampling level necessary for any given objective, cf. Sandefur and Grant (1976), Parker (1978). Sampling grids of various sizes and origins were applied to a simulated map. The evolution and variance of the error of estimation of the mineralized area were then studied, resulting in guidelines for the sequential exploration of a real deposit. Sandefur and Grant did not actually derive a simulation since they had available very closely spaced drillings on existing properties in the Shirley Basin of Wyoming. An important point that these studies highlight is that the use of geostatistics permits preliminary evaluation of the potential mineable reserves in an orebody with much less drilling than is currently practiced in much of the uranium industry.

3) Studies of the sensitivity of resource-reserve estimates to different estimation procedures (including kriging) currently used in the mining industry, cf. Dowd and David (1975), Marechal (1975), and Bernuy and Journel (1977). From a given set of original data, different estimation procedures could lead to quite different estimates. It is thus very important for the investor to check using simulation such theoretical assertions as "Kriging provides the *best* linear estimate." When the deposit is already in production, the check can be made using real data from already mined stopes, cf. the case study of Chuquicamata by Ugarte (1972), which is extensively quoted in Journel and Huijbregts (1978).

4) Studies detailing the sensitivity of the recovery of in-situ geological resources to various parameters, including:

- Size of the selective mining unit, cf. Switzer and Parker (1975). The choice of a given size of mining unit creates restrictions on the choice of the mining method.
- Amount of future information (e.g. number of blastholes) available at the instant of selection, cf. Bernuy and Journel (1977). The real future information is not yet available but can be simulated.
- Clustering of rich and poor patches of ore, cf. Rebollo (1977) and Deraisme (1978.) An isolated rich block may not be recovered because waste intercalated with the rich ore will represent a dilution. The concept of cutoff grade does not by itself take that dilution into account; however, simulation of the spatial pattern of ore vs. waste allows prediction of the dilution factor.
- Contact dilution at hangingwall and footwall, cf. H. Parker (personal communication).

5) Selection of mining method, equipment size, and type of haulage, cf. Clark and White (1977), Guarascio et al (1978), and Deraisme (1977-78). On a given simulated deposit, various alternative methods and equipment choices are tested. Since the simulated deposit is almost perfectly known, the economic and technical consequences of each alternative on a given production period can be calculated. Since a simulation can differ from reality, the calculation procedure should be repeated on various independent simulations of the same deposit. Clark and White show an application of simulation to the critical problem of deciding when and how to switch an open-pit operation to underground. The studies of Guarascio et al, unfortunately not yet available in English, represent the most complete and striking example of simulation of a whole mining operation. The operation simulated was the Pb-Zn mining complex of Masua in Sardinia-Italy. Starting with conditional simulations of the various deposits mined, the whole chain of operations, from blasting, hauling, and blending to mill feeding, has been analyzed, programmed, and simulated. Extension of the simulation to the milling operations is also considered.

6) Studies of blending from various stopes to stabilize tonnage and quality of production, cf. H. Parker (personal communication), Marechal and Shrivastava (1977), and Deraisme (1977).

7) A study of the determination of production rates and cutoff grades that vary in time, and their impact on recovered benefits, cf. Dowd (1976).

8) In nonmining endeavors, two striking applications of Conditional Simulations should be mentioned:

- Simulations of precipitation in desert provinces for the study of water runoff and recovery (Figs. 1 and 2), cf. Delhomme and Delfiner (1973).
- Simulations of the possible fluctuations in the geometry of an oil reservoir around its estimated image to study the

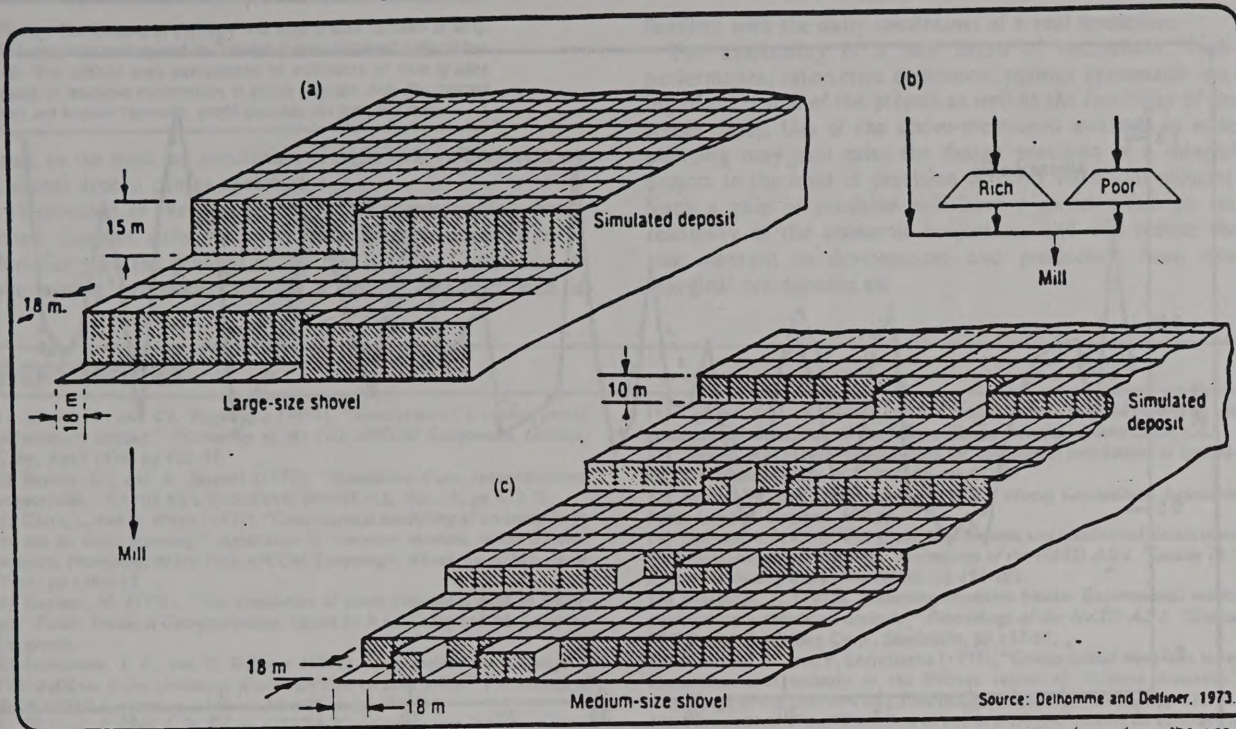


Fig. 3—Three alternative mining and blending processes. In alternative (a), production from large-size shovels is sent directly to the mill. In alternative (b), part of the previous production passes through a stockpile comprised of two subpiles. In alternative (c), on-site blending is performed via the use of numerous medium-size shovels operating on smaller benches.

uncertainty of oil recovery, cf. Delfiner and Chiles (1977).

CASE STUDY—BLENDING OPTIONS

At one porphyry copper open-pit operation, various mining and blending procedures were considered for a particular ore zone. The object was to assess their influence on the fluctuations of the mill feed grades over time. The first alternative was to meet the production requirements using large-sized shovels, with ore being sent directly to the mill (Fig. 3a). The second alternative was to send part of the previous production to stockpiles, where some blending takes place; the daily mill feed would be obtained from ore coming either directly from the blasts or from the piles (Fig. 3b). The third alternative was an on-site blending of mined ore using a greater number of medium-sized shovels working at different locations in the deposit (Fig. 3c). Proceeding from alternative (a) (worst control) to alternative (c) (best control), the more precise the control on mill feed grades, the greater the mining and handling costs.

The source model used in the simulation included a strongly asymmetrical histogram of Cu grades with fixed mean and variance, a model of spatial continuity of grades characterized by an isotropic spherical variogram with nugget effect. The simulation included both the true average grade of a mining unit (block size 18 x 18 x 10 m) and its estimator deduced from a set of blastholes. As in a real operation, selection of ore and waste and mill feed control was based on blasthole estimates rather than on true block grades.

The simulated data base was stored in the memory of an interactive computer, and simulations of the three alternative mining processes were made. The daily decisions, such as which block to mine and whether to send it to the mill or to the stockpile, were made by the mining engineer using a cathode-ray display of the state of the various faces and stockpiles, as well as an awareness of the past and present mill feed grades. The engineer's decision was certainly not always optimal; however, the object of the study was precisely to simulate reality (which includes human errors) instead of achieving an absolute optimum, which is inaccessible in practice. A whole year's production employing each alternative was simulated in just two days of interactive work with the computer. Results for each alternative included:

- A display of the fluctuations of the daily mill feed grades (Fig. 4). The mill operator will obviously prefer alternatives (b) and (c), whereas the mine operator will prefer alternative (a) for its simplicity and lower mining costs.
- An analysis of how the error of estimation of mean block grades influences the blending processes. On Fig. 4c, on-site blending performed on the estimated grades (dashed line) appears almost perfect, but still there are residual daily fluctuations in the true grades received by the mill. To reduce these fluctuations, additional data must be obtained from a denser pattern of blastholes.
- An analysis of how the size and use of the stockpile affects the regulation of daily production in terms of both tonnage and grades. The analysis indicated that a stockpile equal to three or four days' production was sufficient to stabilize mill feed grades over a two-week period.
- An analysis of how different spatial locations of clusters of rich and poor ore affect blending processes and total recovery of ore when selection by cutoff grade is considered.

In conclusion, the Conditional Simulations technique offers the possibility of deriving various simulated deposits from a given source model, which in turn is built from the geological and geostatistical information available on the real deposit. Since each simulation employs the same known data, all simulated deposits have an equal likelihood of being the real deposit. As simulated deposits are almost perfectly known, all exploration or production programs planned on the real deposit can be tested on the simulations and their consequences can be displayed. These planned processes can then be corrected for better efficiency, flexibility, or robust-

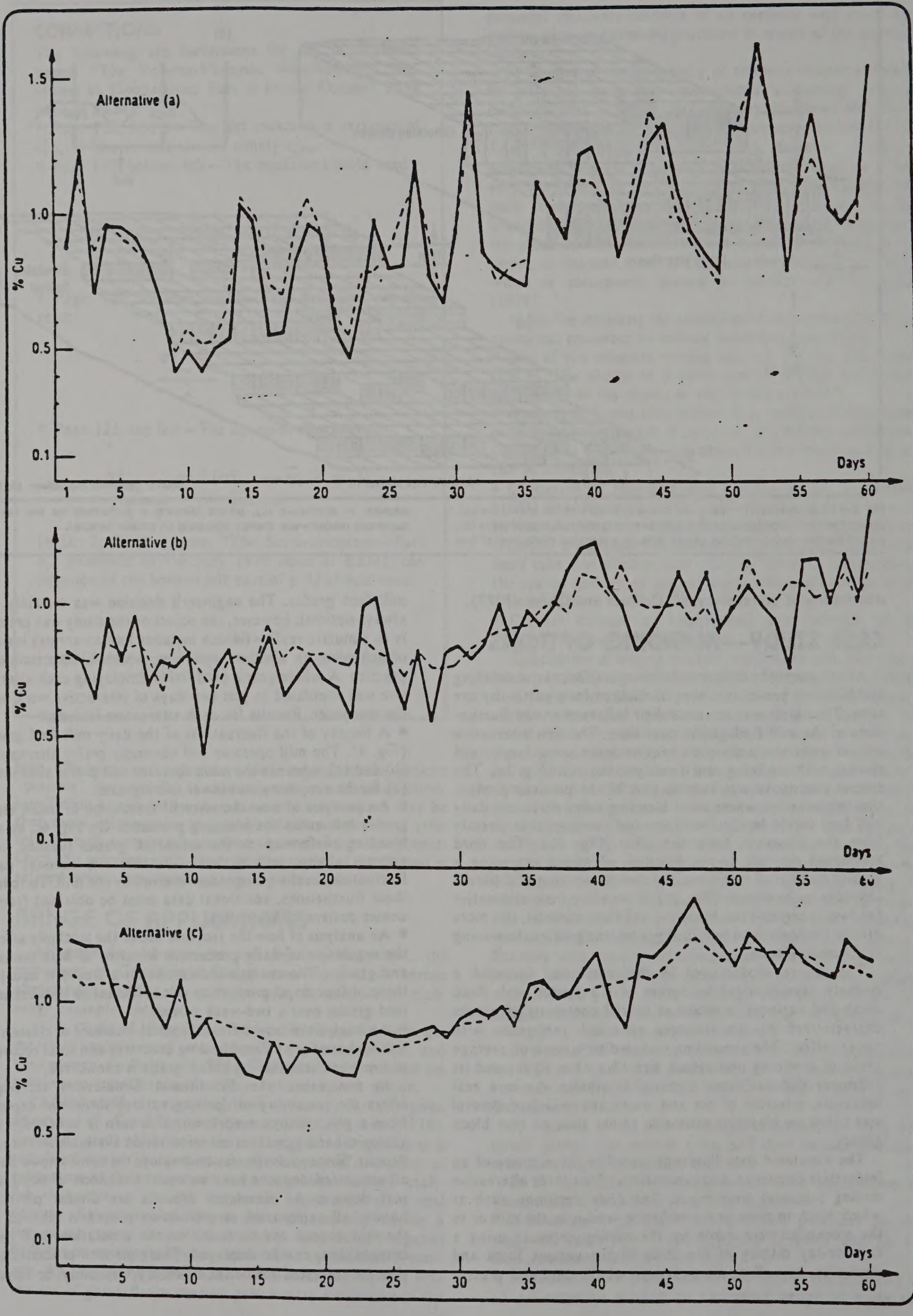

Alternative (a)
% Cu
1.5
1.0
0.5
0.1
Days
1
5
10
15
20
25
30
35
40
45
50
55
60
Alternative (b)
Alternative (c)

Fig. 4—Fluctuations in the daily mill feed grades (shown at left). The solid lines correspond to the true grades received daily by the mill. The dashed lines correspond to estimates of true grades based on blasthole information. In actual practice, only the dashed lines are known; however, profit depends on the solid lines.

ness, or the need for acquisition of additional information on the real deposit can be specified.

Derivation of the source model and corresponding simulated deposits calls for an expert geostatistician who is familiar with the geology of the real deposit. Simulation of the mining processes calls for a mining engineer who is familiar with the daily constraints of a real operation.

The availability of a new breed of inexpensive, high-performance, interactive computers permits systematic testing of every link of the project as well as the flexibility of the whole chain. Use of the above-mentioned methods in mine planning may well raise the design precision of a mineral project to the level of precision of other industrial projects. Such a gain in precision will have a direct effect on the reliability of the economic projections and will reduce the risk inherent in development and production from new marginal ore deposits.■

REFERENCES

1) Alfaro, M., and Ch. Huijbregts (1974), "Simulation of a subhorizontal sedimentary deposit," *Proceedings of the 12th APCOM Symposium*, Golden, Colo., April 1976, pp F65-77.

2) Bernuy, O., and A. Journel (1977), "Simulation d'une reconnaissance sequentielle," REVUE DE L'INDUSTRIE MINERALE, Vol. 59, pp 472-78.

3) Clark, I., and B. White (1977), "Geostatistical modelling of an orebody as an aid to mine planning," *Application of computer methods in the mineral industry, Proceedings of the 14th APCOM Symposium*, edited by AIME, New York, pp 1,004-12.

4) Dagbert, M. (1978), "The simulation of space dependent data in geology," *Future Trends in Geomathematics*, edited by R.G. Craig and M. Labovitz (in press).

5) Delhomme, J. P., and P. Delfiner (1973), "Application du Krigeage a l'optimisation d'une campagne pluviometrique en zone aride," *Proceedings of DEWARPID Symposium*, Madrid, June 1973, pp 191-210.

6) Delfiner, P., and J. P. Chiles (1977), "Conditional simulations: A new Monte Carlo approach to probabilistic evaluation of hydrocarbon in place," Internal report N-526 of the Centre de Geostatistique, Fontainebleau, to be published.

7) Deraisme, J. (1977), "Modelisation de gisements et Choix d'une methode d'exploitation," REVUE DE L'INDUSTRIE MINERALE, Vol. 59, pp 483-89.

8) Deraisme, J. (1978), "Simulations sur modeles de gisement de processus miniers et mineralurgiques," Doctoral Thesis, Ecole Nat. Sup. des Mines de Paris.

9) Dowd, P., and M. David (1975), "Planning from estimates: Sensitivity of mine production schedules to estimation methods," *Proceedings of the NATO A.S.I. "Geostat 75,"* Reidel Publishing Corp., Dordrecht, pp 163-83.

10) Dowd, P. (1976), "Application of dynamic and stochastic programming to optimize cutoff grades and production rates," TRANSACTIONS OF THE IMM, Vol. 85, pp A21-31.

11) Guarascio, M., R. Musso, and R. Dumay, various internal reports on the geostatistical simulation of the Masua Pb-Zn complex, Ammi Sarda Co.

12) Journel, A. (1974), "Geostatistics for conditional simulations of orebodies," ECONOMIC GEOLOGY, Vol. 69, pp 673-87.

13) Journel, A., and Ch. Huijbregts (1978), *Mining Geostatistics*, Academic Press, London, England. 600 pp.

14) Journel, A. (1975), "Ore grade distributions and conditional simulations: Two geostatistical approaches," *Proceedings of the NATO A.S.I. "Geostat 75,"* Reidel Publishing Corp., Dordaecht, pp 195-202.

15) Marechal, A. (1975), "Selecting mineable blocks: Experimental results observed on a simulated orebody," *Proceedings of the NATO A.S.I. "Geostat 75,"* Reidel Publishing Corp., Dordrecht, pp 137-61.

16) Marechal, A., and P. Shrivastava (1977), "Geostatistical study of a lower Proterozoic iron orebody in the Pilbara region of Western Australia," *Proceedings of the 15th APCOM Symposium*, Brisbane, Australia, pp 221-30.

17) Parker, H. (1978), "The effect of exploration grid spacing on accuracy of ore reserve determination for various cut-off criteria," a Fluor Mining and Metals confidential report.

18) Rebollo, T. L. (1977), "Etude de l'exploitation du gisement de Fuentes Rosas a l'aide de la simulation non-conditionnelle de blocs," REVUE DE L'INDUSTRIE MINERALE, Vol. 59, pp 479, 482.

19) Sandefur, R., and D. Grant (1976), "Preliminary evaluation of uranium deposits: A geostatistical study of drilling density in Wyoming solution fronts," *Proceedings of the 1976 IAEA Meeting*, Vienna, Austria.

20) Switzer, P., and H. Parker (1975), "The problem of ore versus waste discrimination for individual blocks: The lognormal model," *Proceedings of the NATO A.S.I. "Geostat 75,"* Reidel Publishing Corp., Dordrecht, pp 203-18.

21) Ugarte, I. (1972), "Ejemplos de modelos de estimacion a corto y large plazo," BOLETIN DE GEOSTADISTICA, Vol. 4, pp 3-22, Universidad de Santiago, Chile.

A CASE STUDY: KRIGING FOR ORE VALUATION AND MINE PLANNING

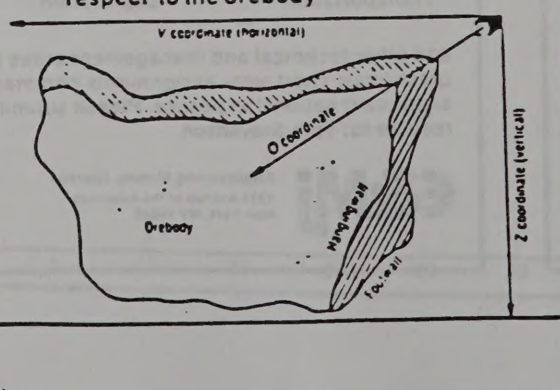
Dr. Jean-Michel Rendu, specialist engineer,
Golder Associates Inc.

The application of kriging methods and computer technology to mineral deposit valuation is gaining momentum because of the increased accuracy and speed of valuation that these techniques afford. While previous papers in this series have dealt with kriging and other geostatistical methods from a theoretical stance, this article details the application of such methods to the problem of reserves estimation in a currently operating mine.* The techniques employed have gained acceptance by staff geologists and stope planners and are considered to be integral elements in the routine reassessment of mining parameters.

The Prieska mine is located in the Cape province of South Africa. The orebody is a massive sulphide copper-zinc deposit having a nearly vertical dip, a strike length exceeding 1.8 km, a depth of more than 800 m, and an average thickness of 9 m. To estimate the grade of the ore, samples from diamond drillholes and chip samples are taken between the footwall and hangingwall. These samples are also used in conjunction with other ore exposures to determine the position of the footwall and hangingwall. The ore is mined using a sublevel open-stope mining method; the stopes created are typically 58 m high x 45 m long along strike. Ore production from the mine exceeds 200,000 tpm.

The use of computerized methods was introduced early in the development stage of the mine.[4] All sample values, sample positions, geologic information, stope limits, and other relevant information are kept on computer files and continuously updated. This information is used in various suites of programs, some of which make extensive use of geostatistical methods. Geostatistical analyses of sample values are repeated whenever warranted by the availability of new data, and random kriging and universal kriging are used to calculate the ore reserves semiannually.

Fig. 1—Position of reference plane ZV with respect to the orebody



METHOD OF RESERVES ESTIMATION

Every six months, the ore reserves remaining in each stope must be recalculated, incorporating any new information that may be available, including possible changes in mining costs and metal prices. The tonnage of ore and average metal content (percentages of copper, zinc, lead, and bismuth) must be calculated for each ore block. To do these calculations, the problem is reduced to two dimensions. A vertical plane parallel to the average strike of the orebody is used as a reference plane (Fig. 1). The point at which a borehole intersects the footwall or hangingwall surface is defined by its distance O from the reference plane, and by its coordinates Z and V in the plane. For each borehole, the average specific gravity and average ore grades are calculated between the footwall and hangingwall. The averages are treated as if they were representative of the orebody at point (Z,V) where Z and V are the coordinates of the center of the borehole-ore intersection. The reduction of the problem to two dimensions is illustrated in Fig. 2. Changes in the mining costs or metal prices may affect the position of the mining footwall and hangingwall, and therefore may affect average grades and average specific gravity.

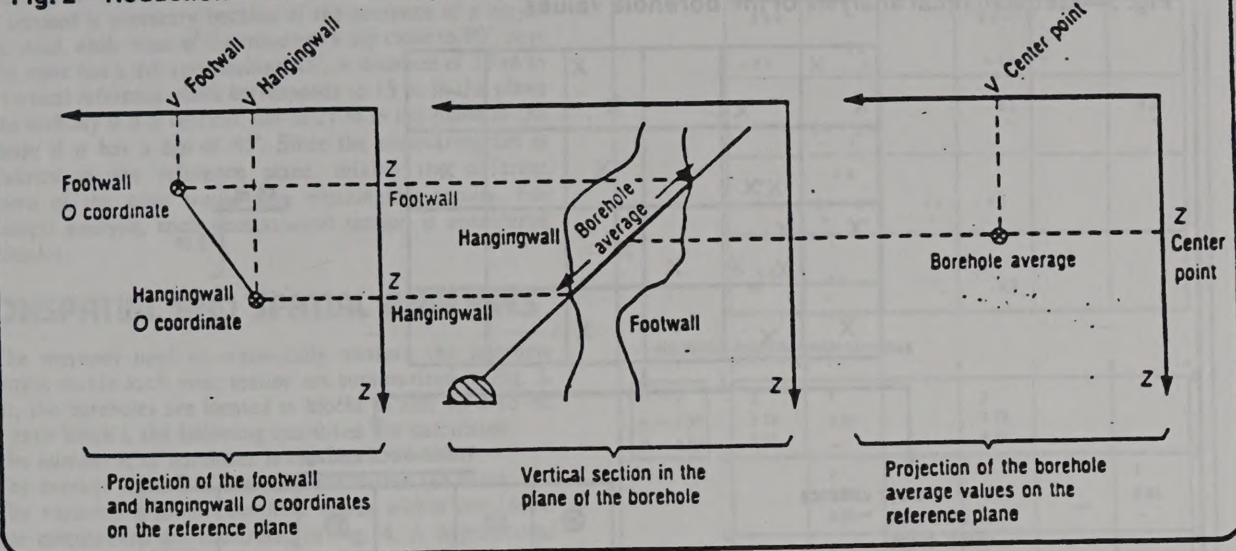
Once the borehole information has been reduced to two dimensions, the following quantities are calculated on a 15 x 15-m grid in the ZV reference plane:

- The distance between the hangingwall surface and the reference plane, using universal kriging.
- The distance between the footwall surface and the reference plane, using universal kriging.
- The average specific gravity and average grades of the 15 x 15-m block centered on the grid point, using random kriging.

From these quantities, the volume, tonnage, and metal

*This article describes geostatistical methods as they were employed in 1977. Many improvements in the techniques have been added in the last two years that are not covered in this report.

Fig. 2—Reduction of the valuation problem to two dimensions



content of each 15 x 15-m block are calculated. The characteristics of the stope being evaluated may be obtained by integrating all of the 15 x 15-m blocks, or those parts that are included in the stope.

CHOICE OF A KRIGING METHOD

The average grades and average specific gravity of each 15 x 15-m block are calculated using random kriging.[1,7,10] This method is preferred to the more classical form of sample kriging because a) the number of boreholes available for valuation is very large, b) the boreholes are not located on a regular grid but the borehole density is fairly constant, and c) the borehole values present a high nugget effect, which implies that the exact borehole position is not significant for valuation purposes. When using random kriging, the position of a borehole within the 15 x 15-m block is ignored; instead, it is assumed that the borehole is randomly located within the block.

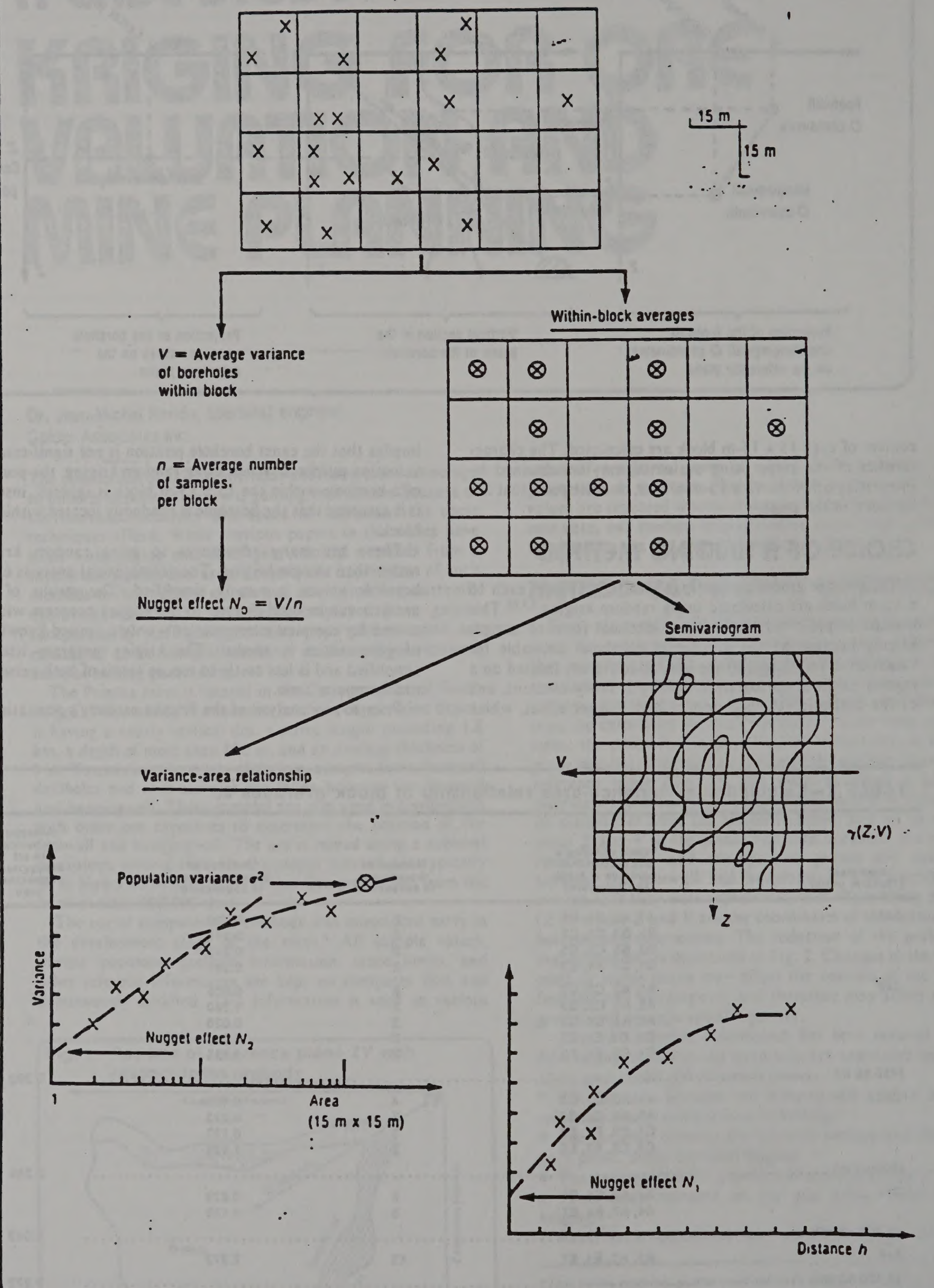
There are many advantages to using random kriging rather than sample kriging. The geostatistical analysis of the borehole values is greatly simplified. The results of this analysis can be used directly in the kriging program without a need for complex interpretation—only a limited knowledge of geostatistics is needed. The kriging program itself is simplified and is less costly to run, in terms of both personnel and computer time.

Prior to any analysis of the Prieska orebody's geostatistical

TABLE 1—Calculation of variance-area relationship of block averages $\overline{x}_i$

Area size (15x15-m units)	Points defining a subsection that contains at least two boreholes	Number of boreholes in subsection	Variance of borehole values in subsection	Average variance in all subsections of specified area
2x1	A1, A3, B1, B3	2	0.320	
	B1, B3, C1, C3	2	0.245	
	B3, B5, C3, C5	2	0.245	
	D1, D3, E1, E3	2	0.245	
1x2	A1, A2, C1, C2	2	0.005	
	A2, A3, C2, C3	2	1.280	
	A4, A5, C4, C5	2	0.020	
	C2, C3, E2, E3	2	0.045	
	C4, C5, E4, E5	2	1.125	
(450 sq m)				0.392
2x2	A1, A3, C1, C3	4	0.429	
	A3, A5, C3, C5	3	0.223	
	C1, C3, E1, E3	3	0.123	
	C3, C5, E3, E5	2	1.125	
(900 sq m)				0.388
3x4	A1, A4, E1, E4	8	0.829	
	A4, A7, E4, E7	5	4.165	
(2,700 sq mi)				2.042
6x4	A1, A7, E1, E7	13	2.272	
(5,400 sq mi)				2 272

Fig. 3—Geostatistical analysis of the borehole values



properties, the mine is divided into contiguous geostatistical sections of fairly constant strike and dip. Division of the mine into sections is necessary because of the presence of a major fault. Also, while most of the mine has a dip close to 90°, part of the mine has a dip approaching 45°. A distance of 15 m in the vertical reference plane corresponds to 15 m in the plane of the orebody if it is vertical, but to 21 m in the plane of the orebody if it has a dip of 45°. Since the semivariogram is calculated in the reference plane, mixing the different sections of the mine would give meaningless results. For statistical analysis, each geostatistical section is considered separately.

NONSPATIAL AND SPATIAL ANALYSES

The methods used to statistically analyze the borehole averages within each mine section are summarized in Fig. 3. First, the boreholes are located in blocks of size 15 x 15 m. For each block i, the following quantities are calculated:

- The number n_i of boreholes in the block.
- The average value $\bar{x}$ of the boreholes within the block.
- The variance v_i of the borehole values within the block.

These calculations are illustrated in Fig. 4. A hypothetical mine section containing 18 boreholes is represented. The section is divided into 24 15 x 15-m blocks, and the quantities n_i, $\bar{x}_i$, and v_i are calculated for each one of the 13 blocks intersected by at least one borehole.

An average within-block variance v is then calculated from the values of v_i. Only the blocks containing at least two boreholes can be used for this calculation. Thus, in Fig. 4 only four blocks are considered:

$$v = \sum_{i=1}^{4} (n_i - 1)v_i / \sum_{i=1}^{4} (n_i - 1)$$

$$v = (1\cdot0.32 + 1\cdot0.32 + 1\cdot0.02 + 2\cdot0.28)/5 = 0.188$$

The average number n of boreholes per block is also calculated, taking into account all the blocks containing at least one borehole:

$$n = 18/13 = 1.38$$

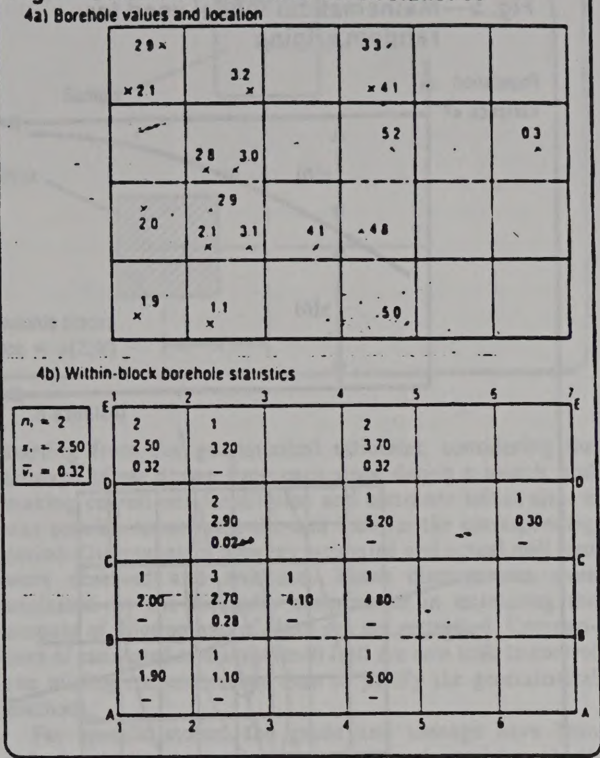
The ratio $N_0 = v/n$ is an estimate of the mean squared error of estimation made when estimating a block value by the average value $\bar{x}_i$ of the boreholes it contains. In other words, N_0 is an estimate of the nugget effect of the within-block borehole averages $\bar{x}_i$.

Aside from the calculation of v, n, and N_0, all other geostatistical analyses (Fig. 3) are performed on the borehole average values $\bar{x}_i$. These values are located on a regular but incomplete grid (Fig. 4-b). The semivariogram of these values is first calculated using standard methods described in earlier articles in this series. A mathematical model $\gamma(Z;V)$ is fitted to the calculated semivariogram, and an estimate N_1 of the nugget effect is obtained:

$$N_1 = \gamma(0;0)$$

Also calculated is the variance of the 15 x 15-m averages $\bar{x}_i$ in areas of increasing size, i.e. the variance-area relationship.[8] For this purpose, the variance of the 15 x 15-m averages are calculated in areas of size 15 x 30 m, 30 x 30 m, 30 x 45 m, 45 x 45 m, etc. An example of this calculation is given in Table 1, where the values of $\bar{x}_i$ obtained in Fig. 4-b are used. From the variance-area relationship, the population variance σ^2 and the nugget effect are estimated. The population variance is estimated by the variance of all the 15 x 15-m averages in the mine section considered (in our example, an

Fig.4—Calculation of within-block statistics



estimate of σ^2 is 2.272). Since the nugget effect is the variance of the 15 x 15-m averages within a 15 x 15-m area, an estimate N_2 is obtained by extrapolation of the variance-area relationship to an area of size 15 x 15 m.

From the above statistical analyses, the following data are retained (Fig. 5):

- The population variance σ^2.
- The nugget effect N. Three estimates—N_0, N_1, and N_2—of N have been obtained. The mathematical models fitted to the observations must be adjusted so that these estimates are reconciled.
- A mathematical model $\gamma(Z;V)$ of the semivariogram. In the solution of the kriging system, the auto-covariogram $\sigma(Z;V) = \sigma^2 - \gamma(Z;V)$ will be used.

THE RANDOM KRIGING PROCESS

Once the statistical analysis of all the sections in the mine has been completed, estimation of the average value of the 15 x 15-m blocks in the sampled area becomes possible. First, the mine is divided in blocks of 15 x 15 m in the ZV reference plane, and the average value of the boreholes within each block is calculated. From that point on, the position of the boreholes within each block is ignored. A sampled block will be referred to hereafter as a "sample," while the average value of the boreholes within a sampled block will be the "sample value." The "position" of a sample is measured at the center of the sampled block, and the distance between two samples is the distance between the two centers. The distance between a sample and the block to be evaluated is the distance between the sample and the center of that block.

To estimate a block, the values of samples in the neighborhood of the block must be found. All the samples within the surrounding peripheries (Fig. 6) are retained. The number of peripheries considered is determined by the borehole density.

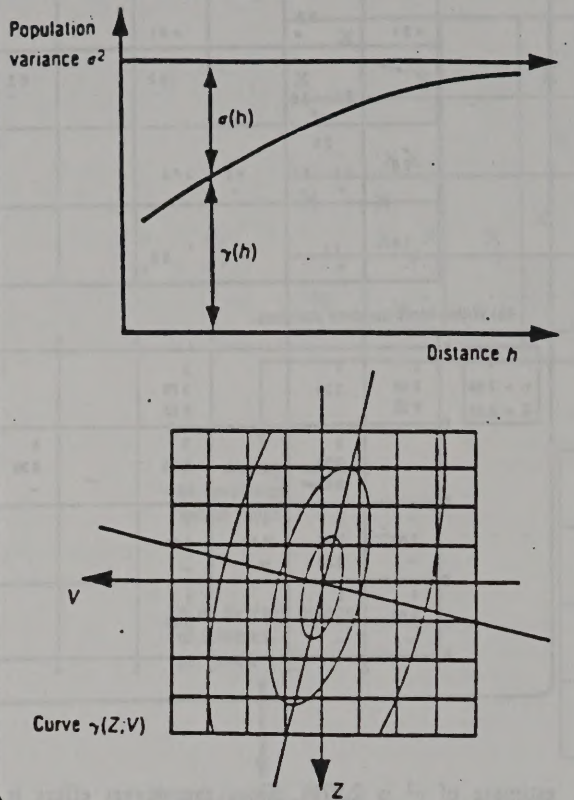


Fig. 5—Mathematical model used for random kriging

To calculate the kriging estimate of the block from the sample values, the following variances and covariances must be calculated:

- The block variance, or variance of the true block value within the mine section, $\sigma^2 - N$.
- The block-to-sample covariance. If the sample is within the block, the block variance $\sigma^2 - N$ is used. If the sample is outside the block, the covariance $\sigma(Z;V)$ is used, where $(Z;V)$ is the vector distance between block and sample (Fig. 7).
- The sample variance, σ^2.
- The sample-to-sample covariance $\sigma(Z;V)$, where $(Z;V)$ is the vector distance between the two samples.

Once these quantities have been calculated, evaluation of the block by kriging is straightforward. The kriging procedure has been described in detail in earlier articles in this series. If n samples are used to evaluate the block, the weight b_i given to the i-th sample value is obtained by solving the following system of $n+1$ equations:

$$\sum_{j=1}^{n} b_j \sigma_{ij} = \sigma_{io} + \lambda \qquad \text{for all } i$$

$$\sum_{i=1}^{n} b_i = 1$$

where σ_{ij} is the covariance between samples i and j at distance $(Z_{ij};V_{ij})$, σ_{io} is the covariance between sample i and the block, and λ is a Lagrange multiplier. Note that:

$$\sigma_{ij} = \sigma\ (Z_{ij};V_{ij}) \text{ if } i \neq j$$
$$\sigma_{ij} = \sigma^2 \qquad \text{if } i = j$$

and if $(Z_{io};V_{io})$ is the distance between sample i and the block, then:

$$\sigma_{io} = \sigma\ (Z_{io};V_{io}) \text{ if sample } i \text{ is outside the block}$$
$$\sigma_{io} = \sigma^2 - N \text{ if sample } i \text{ is inside the block}$$

The kriging error of estimation of the block is:

$$\sigma_K^2 = \sigma^2 - N - \sum_{i=1}^{n} b_i \sigma_{io} + \lambda$$

UNIVERSAL KRIGING

To estimate the volume of ore in a 15 x 15-m block, the positions of the footwall and hangingwall surfaces are estimated relative to the center of the block. A point on these surfaces is defined by its O, Z, and V coordinates, the O

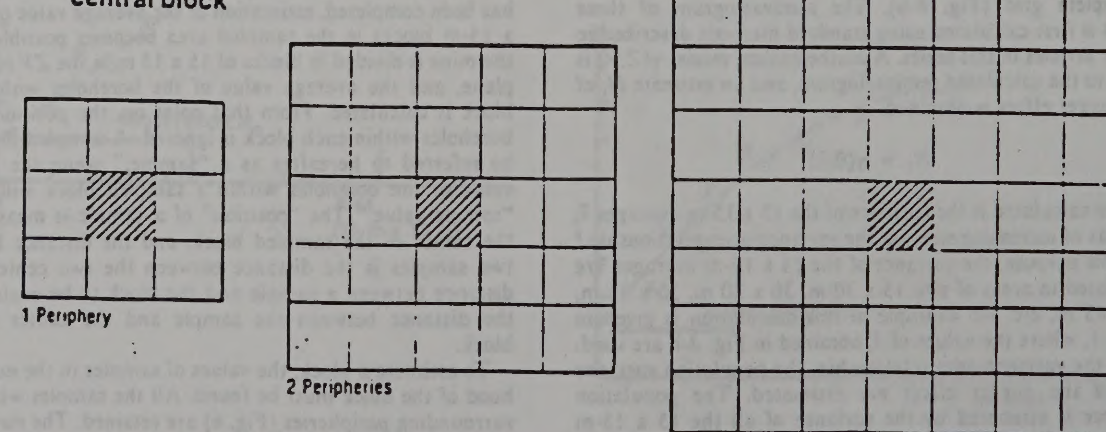
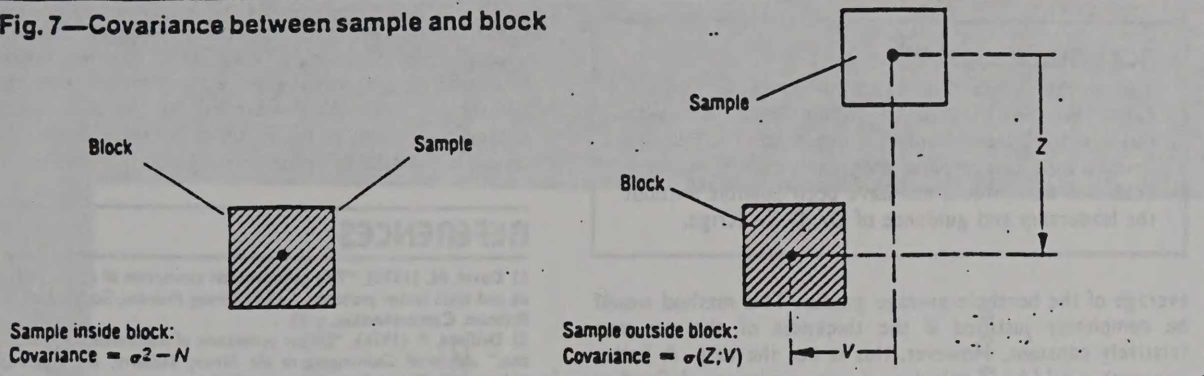


Fig. 6—Blocks considered in valuing the 15 x 15-m central block

Fig. 7—Covariance between sample and block



coordinate being the distance from the ZV reference plane (Fig. 2).

The distance O is treated as the dependent variable, while Z and V are the independent variables. The footwall and hangingwall surfaces are considered as realizations of random functions. These functions are not stationary—the O coordinate is a complex, usually increasing function of Z. Also, the estimated surface must go through the data points—the point semivariogram has no nugget effect and the exact borehole location must be retained. For these reasons, a simplified version of universal kriging is used. To calculate the value of the O coordinate at the center of a block, a plane is first fitted to the closest data points using the generalized least squares method. The center point is then given an O value equal to the value of the plane plus a corrective term obtained by kriging the residual distances between the data points and the plane. An exponential semivariogram with zero nugget effect is used. A detailed description of the method would go beyond the scope of this publication, and can be found in the literature.[6,9,10]

Note that the difference between the estimated hangingwall and footwall O coordinates at the center of a block is an estimate of the horizontal thickness of the orebody at that point, not an estimate of the average thickness of the block. Changes in the computer program are being considered to take this factor into account, but they are not expected to result in very significant improvements.

FOLLOW-UP STUDIES

To ensure that a theoretically correct method such as kriging is accepted by the staff and management of the mine for which it is designed, it is important to prove that the results are at least as good as those obtained by a more traditional method. Unfortunately, comparing calculated ore reserves with the true characteristics of the ore underground is usually impossible, as the latter remains unknown even after mining has been completed. Also, at the Prieska mine, an additional consideration prevails—many of the geostatistical methods were introduced at the very beginning of the mining operation, hence comparison with pre-existing methods was not possible. However, the computer methods were well accepted because they significantly simplified work that would otherwise have to be done manually by the mine staff.

Despite the problems in making direct comparisons, various indirect comparisons of actual and predicted values have been completed. The geostatistical methods described in this paper are used to predict the ore values in situ. The only other reliable independent estimate of the ore values is obtained from the surface plant, where tonnage and grade of the mill feed are known with considerable precision. By starting from the geostatistical estimates, considering the tonnage of ore drawn from each stope during a month, and making corrections for dilution and clean ore left in situ, it was possible to estimate the mill feed for the corresponding period. Discrepancies between estimated and actual mill feed were observed and analyzed. These discrepancies were explained by the difficulty encountered in estimating the amount of dilution and of clean ore not recovered. Comparisons of estimated and actual mill feed are now used to control the mining method rather than to justify the geostatistical methods.

For specific stopes, the grade and tonnage have been calculated using both geostatistical and polynomial methods. Any significant differences observed between methods were assignable to the superior ability of geostatistical methods to decrease the influence of extremely high or extremely low values. Discussion with the mine geologists about these differences invariably resulted in acceptance of the geostatistical estimate as the "better" estimate.

The computer-generated footwall and hangingwall surfaces have been compared with manually drawn surfaces on geologic sections. In most instances, the two surfaces matched perfectly. Most of the discrepancies were explained by one of two factors: Either the geologist possessed information not yet available to the computer, or no reliable information was available. These discrepancies were not due to weaknesses in the geostatistical method.

Despite the somewhat unsatisfactory nature of these follow-up comparisons, they resulted in the general acceptance of the computer methods by the mine staff and management.

WEAKNESSES AND IMPROVEMENTS

Application of various forms of kriging to the Prieska mine has proven to be successful, and the computer programs developed for ore reserves calculation and stope planning are now used routinely. However, many possible improvements are being considered:

- In the random kriging method currently used, the average value of the boreholes in a 15 x 15-m block is given the same weight, whether it is obtained from one or more boreholes. Also ignored is the likely presence of a proportional effect, which would indicate that higher grade values have a higher variance than lower grade values. An improved evaluation would be obtained if the variance of the borehole averages was treated as a function of the number of boreholes and their average value, instead of being considered as a constant.[1] Examples detailing the implementation of the logarithmic kriging method of grade evaluation are on record.[5, 11]
- The average grade of a block is calculated as a weighted

ACKNOWLEDGEMENT

The author is grateful to Anglo-Transvaal Consolidated Investment Co. for the authorization to publish this article. Development and implementation of the Prieska suite of computer programs, part of which is described here, would not have been possible without the leadership and guidance of Dr. D. G. Krige.

average of the borehole average grades. This method would be completely justified if the thickness of the ore was relatively constant. However, this is not the case. A better approach would be to calculate the *accumulations*, defined as the product of the grade times specific gravity and thickness. Originally this could not be done, due to the difficulty in defining the thickness of the ore. Due to the satisfactory results obtained with universal kriging, improvements in this method are now being considered.

■ In some unusual situations related to the location of the surface exposures, the surface contouring program currently used gives unacceptable results. In these instances, substitution of the generalized increments method for the universal kriging method may give better results.[2]

■ Some improvements are expected when the average thickness of a 15 x 15-m block is estimated by the difference between the average hangingwall and footwall *O* coordinates, rather than the difference between the hangingwall and footwall *O* coordinates at the center of the block.

■ Finally, one serious problem has been encountered in the use of an integrated computerized system for ore valuation and mine planning. Mine geologists often possess information about the orebody that for various reasons is not available to the computer. The geologist should thus be able to override the computer results without having to return to entirely manual operations. This could best be done by introducing the use of interactive graphics. The main advantage of this procedure would be that the geologist or stope planner could communicate with the computer without requiring any special computer training.[3] At Prieska, however, introduction of interactive graphics is expected to present serious technical problems due to the remoteness of the mine site, which creates communication difficulties between the mine and the computer located in Johannesburg. ■

REFERENCES

1) David, M. (1970), "The geostatistical estimation of porphyry type deposits and scale factor problems," *1970 Mining Pribram*, Science and Technique Pribram, Czechoslovakia, p 20.

2) Delfiner, P. (1976), "Linear estimation of nonstationary spatial phenomena," *Advanced Geostatistics in the Mining Industry*, M. Guarascio et al. (editors), D. Reidel Publishing, pp 49-68.

3) Dowd, P. A. (1975), "Mine planning and ore reserve estimation with the aid of a digigraphic console display," CANADIAN INSTITUTE OF MINING AND METALLURGY BULLETIN, February, pp 39-43.

4) Krige, D. G. (1973), "Computer applications in investment analysis, ore valuation and planning for the Prieska copper mine," *11th Symposium on Computer Applications in the Mineral Industries*, Tucson, Ariz., April, University of Arizona, pp G31-47.

5) Krige, D. G. (1978), *Lognormal deWijsian Geostatistics for Ore Evaluation*, G.S. Baker (editor), South African Institute of Mining and Metallurgy, 50 pp.

6) Krige, D. G., and J. M. Rendu (1975), "The fitting of contour surfaces to hanging and footwall data for an irregular orebody," *13th Symposium on Computer Applications in the Mineral Industries*, Institut fur Bergbaukunde, Technische Universitat, Clausthal-Zellerfeld, West Germany, October, pp CV1-12.

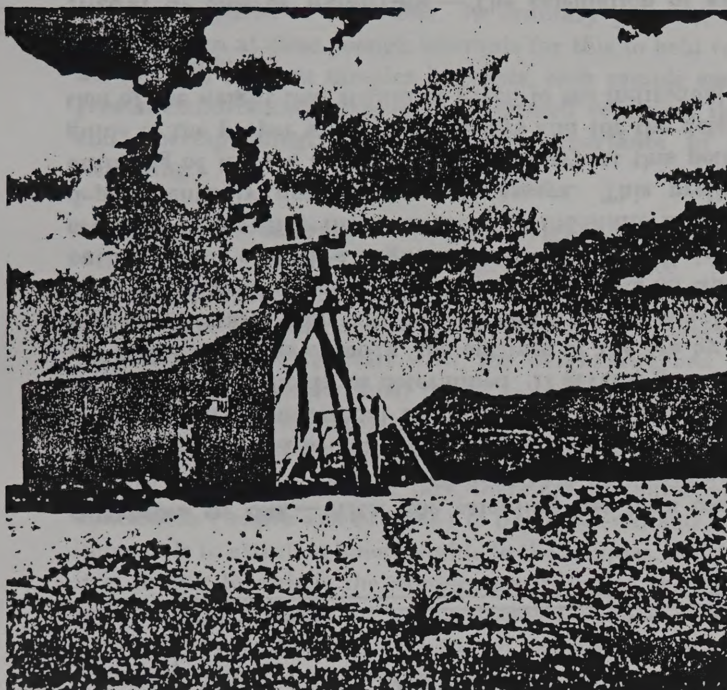
7) Marechal, A., and J. Serra (1970), "Random kriging," *Geostatistics*, D. F. Merriam (editor), Plenum Press, pp 91-112.

8) Rendu, J. M. (1976), "The optimization of sample spacing in South African gold mines," JOURNAL OF THE SOUTH AFRICAN INSTITUTE OF MINING AND METALLURGY, April, pp 392-399.

9) Rendu, J. M. (1976), "A computer package for improved planning of blast hole layout at Prieska Copper Mine," *2nd IFAC Symposium on Automation in Mining, Mineral and Metal Processing*, Johannesburg, South Africa, September, pp T5A1-T5A11.

10) Rendu, J. M. (1978), *An Introduction to Geostatistical Methods of Mineral Evaluation*, G.S. Baker (editor), South African Institute of Mining and Metallurgy, 84 pp.

11) Rendu, J. M. (1979), "Normal and lognormal estimation," JOURNAL OF THE INTERNATIONAL ASSOCIATION FOR MATHEMATICAL GEOLOGY, October, pp 407-422.



Prospect Shaft.

EXAMINATION AND VALUATION OF MINERAL PROPERTY

BAXTER AND PARKS

Fourth Edition 1957

by

ROLAND D. PARKS

Associate Professor of Mineral Industry
Massachusetts Institute of Technology

•

OIL PROPERTY VALUATION

By WALTER L. WHITEHEAD

Associate Professor of Geology
Massachusetts Institute of Technology
in collaboration with the author

THE MICHIGAN MINE APPRAISAL SYSTEM

By FRANKLIN G. PARDEE

formerly
Mining Engineer and Appraiser of Mines
Michigan Geological Survey
Lansing, Michigan

ADDISON–WESLEY PUBLISHING COMPANY, INC.
READING, MASSACHUSETTS

CHAPTER 5

SAMPLING CALCULATIONS

OUTLINING OF ORE — After the sampling has been completed and the analyses have been received from the laboratory, and before attempting to calculate the averages from groups of individual samples, the preliminary outlines of the orebodies should be determined. If several grades of ore are included in one orebody, the outlines of the ore by grades may be necessary. This may be done best from the sample maps, on which should be noted also all geological data on which the outline of the orebodies might depend. The boundaries of the ore may be property lines, contacts, or indefinite limits which depend entirely upon the sample assays. This preliminary outlining of the ore and grades is necessary at this point as a guide to the proper grouping of assays and for the determination of the weight that should be given to the individual assays of each group.

THEORY OF SAMPLE WEIGHTING — The calculation of averages from groups of individual sample assays is a problem of mathematical weighting. Fundamentally, the calculation is based upon a determination of the relative importance of each sample with respect to the other samples with which it is combined. A fairly taken sample is intended to be representative of the orebody or mineral mass at the point where it was taken. In practice, however, there is always an interval or distance spacing between the points tested and, therefore, each sample must be considered as repre-

sentative of a certain surrounding volume, the size of which depends upon the location of the sample in question and upon its relation to adjacent samples or to ore limits.

Thus, when an orebody is systematically sampled at numerous points, each sample must be considered as representative of the ore not only at the point where it is taken, but outward in all directions from this point halfway to all adjacent sample locations. In other words, a sample must continue to be considered as representative of the ore until it is relieved of the responsibility by another sample. This can be shown best by citing the two extreme cases, as follows:

1. If the entire orebody is taken as a sample, the sample, of course, represents the exact assay of the ore, and each sample particle is held only to its own assay because there are no extra particles participating. The interval between points sampled is zero, and the influence of each sample particle is limited to its own volume.
2. If only one point in the entire orebody is sampled, the assay of the one sample must be assumed to represent the whole mass, and the influence of the sample must continue outward from the point tested to the ore limits in all directions. The volume attributable to the sample is the volume of the whole orebody.

In the first of these instances the relative value of each sample particle is the same as that of each other particle and the relative importance of each and all of the particles may be considered as unity, whereas in the second instance the value of the sample particle is a maximum and the relative importance may be considered as infinite.

Thus, the importance of sample weighting varies inversely with the number of samples taken for a given orebody or portion thereof. If a large enough number of samples is taken, a straight unweighted average will give a true result because the intervals between samples are correspondingly reduced and each sample is therefore representative of an insignificant vol-

ume as compared to the whole. In ordinary practice, samples are not taken at close enough intervals for this to hold true. As the interval between samples increases, each sample assumes a greater importance because of its increased volume of influence and, correspondingly, increases the importance of sample weighting.

WEIGHTING OF SAMPLES — The true weight of a sample is its ''volume of influence''; hence, whenever feasible, sample weights should be assigned in ratio to the respective volumes. In grouping samples where three dimensions are being dealt with, the weight attached to each sample must be proportional to the volume of the block of ore at whose center the sample was taken.

In determining the assay value of a plane or warped surface such as a stope face or drift face, there are, of course, only two dimensions (length and breadth) to be considered and it will be impossible to attach volume weights to the samples. In this case, it is proper to weight the samples according to the ''areas of influence'' at whose centers the respective samples were taken.

Again, in the instance of a row of samples, where only one dimension (linear) is involved, it is appropriate to consider the ''distance of influence'' of each sample as its proper weight.

It must be borne in mind, however, that the ultimate aim in sampling is to determine the value of a volume of ore. Likewise, the ultimate purpose of weighting each sample that enters into the calculation of an average is to attribute to the sample its respective volume.

Although, in certain instances, weights may be computed on distance between samples (linear dimension only) or on areas of influence (two dimensions only), these are but preliminary steps toward the determination of true volume weights and are used either because the volume cannot be computed for lack of data or because the areas or distances are in proportion to volume.

When samples are taken equidistant from each other in one

dimension, the interval distance may be omitted from the calculations because it would factor out if included; when samples are taken equidistant from each other in two dimensions, both of these distances may be omitted from the calculations for the same reason; and if equal-sized samples are taken equidistant from each other in each of three dimensions, the samples may all be mixed to form a single composite sample because the dimensional intervals, if included, merely impute identical volumes to each sample.

DEGREE OF ACCURACY — The subject of ''assay weighting,'' though discussed at length in the transactions of most of the mining societies and in mining literature in general, remains a debatable question when attempting an exact mathematical analysis. Fortunately, the refinements which have entailed the greatest amount of controversy are, in a sense, beyond the limits of practical application. Except by chance, a sample cannot be exact; in grouping sample assays, it is wasted effort to attempt a calculation that will be mathematically rigid, when the data upon which the calculation is based probably contain certain inherent errors. In all sampling calculations the probable degree of accuracy attainable should be borne in mind, so that the various phases of the calculations will be kept precise, relative to each other. The important point is that while being combined, the assays must be logically weighted according to what they represent. Any deviations from this plan must be based upon sound reasoning.

SPACING AND ARRANGEMENT OF SAMPLES — The purpose of this text in dealing with sampling calculations is to set forth the fundamental principles involved in grouping assays. Practically all sampling estimates are individual problems in some respect or another, but the theory remains the same. The following examples illustrate the theory; no attempt is made, however, to cover all possible cases.

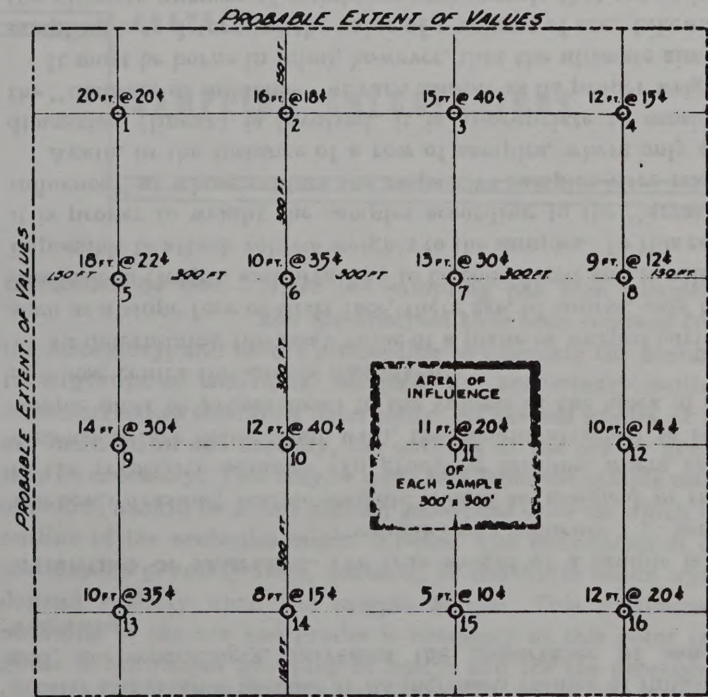


Fig. 3. Placer area sampled at regular intervals.

UNIFORM SPACING ON RECTANGULAR CO-ORDINATES

EXAMPLE 1.

Extended Area Fig. 3 is the plan assay map of a placer area which has been drilled and sampled at each 300-ft. intersection of a rectangular co-ordinate system. It is assumed that there are no property, assay, or other boundaries which limit the extent of the deposit, and that values extend one-half of the co-ordinate interval beyond the outside rows on each side. Each drill hole will lie at the center of a 300-ft. square of influence.

It will be apparent that, so far as area is concerned, each sample has equal influence and that if the depth (thickness) of the placer were equal at all intersections, a straight average of the sample assays would give the average analysis of the placer; but, in considering volume, the samples are not of equal importance, as the depths of the sampled holes vary. The samples are, therefore, weighted according to depth by taking the assay-foot product. Though not so stated, this gives a volume weight to each sample. The computations are as follows:

Sample Lengths		Assay Value		Foot-Assay Product
20 ft.	@	20¢ per cu. yd.		400
16	@	18	—	288
15	@	40	—	600
12	@	15	—	180
18	@	22	—	396
10	@	35	—	350
13	@	30	—	390
9	@	12	—	108
14	@	30	—	420
12	@	40	—	480
11	@	20	—	220
10	@	14	—	140
10	@	35	—	350
8	@	15	—	120
5	@	10	—	50
12	@	20	—	240
195				4732

$4732 \div 195$ — 24.26¢, average assay value per cubic yard.
$195 \div 16$ — 12.18 ft., average thickness of bed.
— 4.06 yd., thickness of bed.

As the ore values are assumed to extend one-half the co-ordinate interval beyond the outside rows of holes, the total area of probable values is a square 400 yards on a side, and therefore:

$400 \times 400 \times 4.06$ — 649,600 cu. yd. of gravel available.
$649{,}600 \times \$.2426$ — \$157,600, total values available.

EXAMPLE 2.

Included Area — Valence Weights If, in Example 1, the volume to be considered were limited to the area bounded by the drill holes, the situation would be as is shown in Fig. 4, where all samples do not have equal areas of influence. The average analysis of each volume inclosed between adjacent co-ordinate lines may be obtained by combining the assay-foot products of the four corners; the average depth of each unit is the arithmetical mean of the four corner-depths; and the several units may then be combined by the use of the depth-assay products, as tabulated on page 52.

It is apparent, however, in the calculation that the corner samples (No. 1, 4, 13, & 16) have entered into the computations only once, the intermediate side samples (No. 2, 3, 5, 8, 9, 12, 14, & 15) twice, and each of the interior samples (No. 6, 7, 10, & 11) four times. Therefore, the entire group of samples may be combined in a single operation by using weights of one,

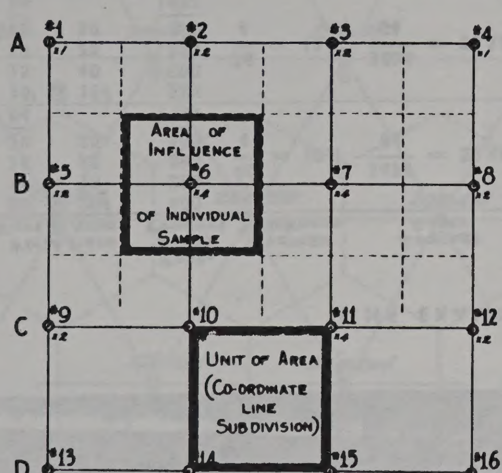


Fig. 4. Sampling at intersections of regular co-ordinates.

two, and four. This weighting is determined by the number of units — in this case squares — affected by each sample. This type of weighting has been aptly termed the "valence" of the sample, from the chemical term denoting the combining power of the elements.

Sample No.	Field Data Feet & Assay		Foot-Assay Product	Unit Areas Affected (Valence)	Valence-Foot Wt.	Valence-Foot-Assay Product
1	20′	@ 20¢	400	1	20	400
2	16	18	288	2	32	576
3	15	40	600	2	30	1200
4	12	15	180	1	12	180
5	18	22	396	2	36	792
6	10	35	350	4	40	1400
7	13	30	390	4	52	1560
8	9	12	108	2	18	216
9	14	30	420	2	28	840
10	12	40	480	4	48	1920
11	11	20	220	4	44	880
12	10	14	140	2	20	280
13	10	35	350	1	10	350
14	8	15	120	2	16	240
15	5	10	50	2	10	100
16	12	20	240	1	12	240
				36	428	11174

11,174 ÷ 428 — 26.11¢, average assay.
428 ÷ 36 — 11.89 ft., average thickness.

The same method may be used in determining the total volume or number of cubic yards in the deposit; i.e., each depth is given a valence or weight in proportion to the number of squares affected. It is apparent that this method of determining the volume is not mathematically exact unless the orebody is flat-lying and the depths are equal. Where conditions warrant, either a dip-factor or the prismoidal formula may be used to obtain the correct volume from the apparent depths shown by drilling. The computations are:

Unit Area No.	Samples No.	Field Data Feet	& Assay	Foot-Assay Product	Average Thickness	Average Assay	Volume-Assay Product (Thickness × Assay)
1	1	20′ @	20¢	400			
	2	16	18	288			
	5	18	22	396	$\frac{64}{4} = 16.0'$	$\frac{1434}{64} = 22.406¢$	358.496
	6	10	35	350			
		64		1434			
2	2	16′ @	18¢	288			
	3	15	40	600			
	6	10	35	350	$\frac{54}{4} = 13.5'$	$\frac{1628}{54} = 30.148¢$	406.998
	7	13	30	390			
		54		1628			
3	3	15′ @	40¢	600			
	4	12	15	180			
	7	13	30	390	$\frac{49}{4} = 12.25'$	$\frac{1278}{49} = 26.082¢$	319.505
	8	9	12	108			
		49		1278			
4	5	18′ @	22¢	396			
	6	10	35	350			
	9	14	30	420	$\frac{54}{4} = 13.5'$	$\frac{1646}{54} = 30.481¢$	411.498
	10	12	40	480			
		54		1646			
5	6	10′ @	35¢	350			
	7	13	30	390			
	10	12	40	480	$\frac{46}{4} = 11.5'$	$\frac{1440}{46} = 31.304¢$	359.996
	11	11	20	220			
		46		1440			
6	7	13′ @	30¢	390			
	8	9	12	108			
	11	11	20	220	$\frac{43}{4} = 10.75'$	$\frac{858}{43} = 19.953¢$	214.496
	12	10	14	140			
		43		858			
7	9	14′ @	30¢	420			
	10	12	40	480			
	13	10	35	350	$\frac{44}{4} = 11.0'$	$\frac{1370}{44} = 31.136¢$	342.496
	14	8	15	120			
		44		1370			
8	10	12′ @	40¢	480			
	11	11	20	220			
	14	8	15	120	$\frac{36}{4} = 9.0'$	$\frac{870}{36} = 24.167¢$	217.50[illegible]
	15	5	10	50			
		36		870			
9	11	11′ @	20¢	220			
	12	10	14	140			
	15	5	10	50	$\frac{38}{4} = 9.5'$	$\frac{650}{38} = 17.105¢$	162.4[illegible]
	16	12	20	240			
		38		650			
					107.00		2793.480

2793.480 ÷ 107.00 = 26.11¢, average assay value per cu yd

107.00 ÷ 9 = 11.89 ft, average thickness

Grouping into Rows The composite assay of the group may also be calculated by averaging each row of holes and then combining the several rows.[1] Since the distance between holes in any row (Fig. 4) is the same throughout the length of the row, the end samples should be given only half the weight of the intermediate samples when combining their foot-assay products. Likewise, when combining the row averages by their depth-assay products to obtain an average for the entire area, the end rows will carry only half the weight of the intermediate rows. If the weight attributable to the corner samples is taken as one, the intermediate side samples will again have weights of two, and the interior samples will again carry weights of four.

The final average result as to assay or thickness for a group of samples taken at the intersections of a regular co-ordinate system will be the same whether the averaging process is viewed from the standpoint of area of influence of individual samples, or of separation into unit areas, or of grouping by rows. In certain instances, when a definite allocation of values is desired, one of these methods would be used to the exclusion of the rest.

Equidistant Spacing on 60° Co-ordinates — Regularly spaced samples need not be on a rectangular co-ordinate system. Samples may be taken at the intersections of a 60° co-ordinate system, as shown in Fig. 5. The unit of area in this case is an equilateral triangle, and the areas of influence of all the points sampled are identical hexagons.

Semi-irregular Spacing — The samples may be regularly or irregularly spaced in rows, and the rows themselves may be regularly or irregularly spaced. In any combination of samples in rows, the average analysis of the plane or section through each row may be determined by weighing the samples in the row by their

---

[1] "Valuation of Dredging Ground"—C. S. Herzig.

respective lengths and distances of influence. The row sections may then be combined to obtain an overall average by weighting each section in proportion to its area and distance of influence. In other words, the samples in a row may be combined by areas of influence to obtain the average for the full section represented by the row and the rows may then be grouped in proportion to their volumes of influence. Fig. 6 shows an example of regularly spaced samples in irregularly spaced rows.

Irregular Spacing — The exploratory drilling of an orebody is seldom done in a consistently systematic manner, for a com-

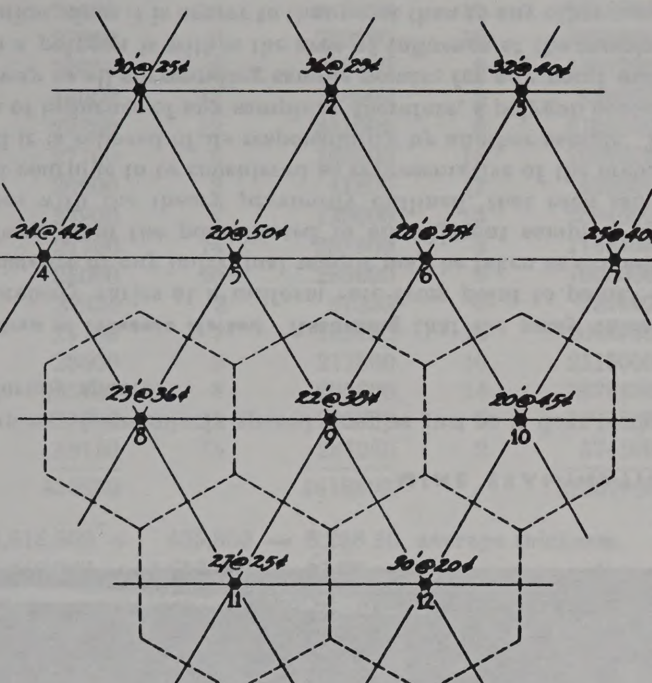


Fig. 5. Samples equidistant from each other on 60° co-ordinates.

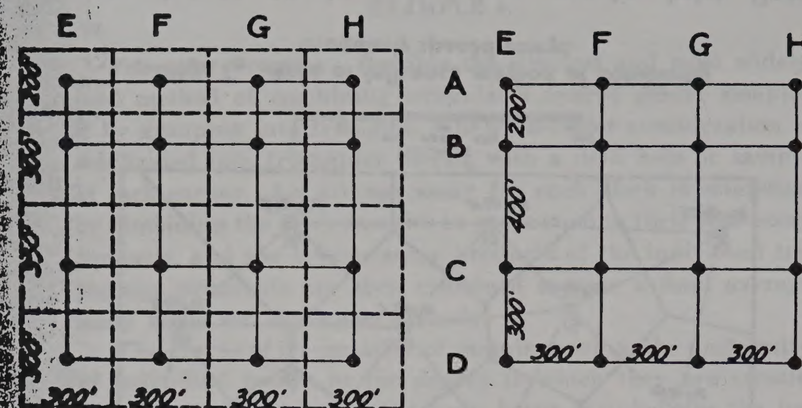


Fig. 6. Sampling at semiregular intervals.

plete program is justified only when preliminary work has roughly outlined the limits of the deposit and shown sufficient promise of values to warrant the necessary expenditure of capital. The engineer sent to examine a property will usually find that the previous work has resulted in a set of irregularly spaced samples, and unless he feels that the work should be done over, his best course is to adapt his program to the results at hand. Often this will mean a continuation of irregular drilling or sampling, with the locations determined by a study of the previous data. A clear understanding of the geology, in particular the structural relationship, may be critical to the effective planning of such a program.

When proper judgment is exercised in combining the several assays, a group of irregularly spaced samples which have been fairly taken should give just as accurate a composite result as samples in a symmetrical layout. Care must be taken to avoid giving undue prominence to rich spots by concentrating too many holes in such areas. The basic principle of weighting each sample according to its volume of influence applies to the

grouping of irregularly spaced samples just as it does to those uniformly spaced.

Area of Influence Method Assuming that the assay value of an orebody varies at a uniform rate from point to point,[1] the persistence of any individual sample may be taken as extending halfway from the point tested to all adjacent samples. This agrees with the theory previously outlined, that each sample must continue to be considered as representative of the orebody until it is relieved of its responsibility by another sample. The area of influence of any sample is, therefore, a polygon bounded midway to all surrounding sample points; for any point within such a polygon is within the area of influence of the sample in question, since it is nearer to that point than to any other sample point. In combining the group of assays, each sample is given its volume-of-influence weight by multiplying its area of influence by both the sample length (thickness or depth of bed) and the assay value to give a volume-assay product.

EXAMPLE 3.

To demonstrate the application of the method, take the hypothetical drilling plan shown in Fig. 7, in which the actual spacing of the holes (sample points) and their respective values are given for an area one-eighth mile square.

The problem is most readily solved graphically with the aid of a planimeter for scaling the respective platted areas, although as an alternative method each polygon may be subdivided into triangles and the area of each triangle computed from base and altitude measurements. The area of influence of any hole — for example, interior hole No. 10 — is defined by drawing the lines 10-5, 10-7, 10-11, 10-14, 10-13, and 10-9, connecting hole No. 10 with each of the surrounding holes. Upon these radial lines the respective perpendicular bisectors AB, BC, CD, DE, EF, and FA are then erected to form the polygon ABCDEF, which bounds the area of influence of the hole in question. This process is then repeated for each of the other holes within the given area to produce a series of polygons such as are shown on the sketch. The respective areas of influence may then be scaled off with a planimeter accurately enough for most purposes. A tabulation of the data and computed results is given on page 58.

[1] "Mining Engineers' Handbook"—R. Peele. John Wiley & Sons, Inc., [illegible] Ed., p. 475.

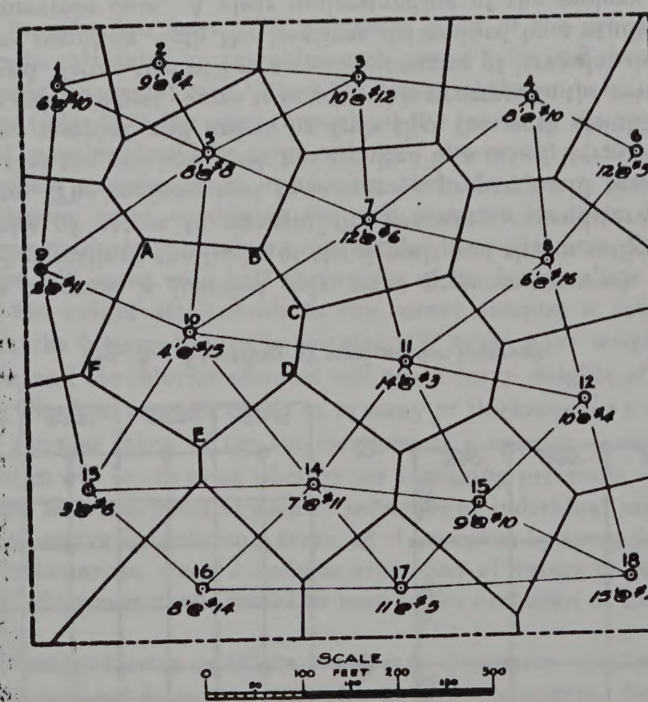


Fig. 7. Area of influence method of combining irregularly spaced assays.

It is readily seen from an inspection of the plan sketch that polygons 13 and 16 are more likely than any of the others to be in error for lack of definite data. An additional drill hole located near the lower left-hand corner of the plat would either assure or disprove the presence of the wide variations in thickness and value that are now apparent between holes 13 and 16.

Hole No.	Area of Influence	Sample Length (Depth) (Thickness)	Volume Factor	Assay	Volume-Assay Product
1	15450	6′	92700	$10	927000
2	13900	9	125100	4	500400
3	24600	10	246000	12	2952000
4	27900	8	223200	10	2232000
5	26050	8	208400	8	1667200
6	14000	12	168000	5	840000
7	29250	12	351000	6	2106000
8	27700	6	166200	16	2659200
9	20900	2	41800	11	459800
10	34900	4	139600	15	2094000
11	27700	14	387800	3	1163400
12	26000	10	260000	4	1040000
13	32450	3	97350	6	584100
14	27100	7	189700	11	2086700
15	23500	9	211500	10	2115000
16	25700	8	205600	14	2878400
17	19600	11	215600	5	1078000
18	19150	15	287250	2	574500
	435850		3616800		27957700

3,616,800 ÷ 435,850 — 8.298 ft., average thickness.
27,957,700 ÷ 3,616,800 — $7.73 per ton, average value.

Assuming the specific gravity of the ore to be 3.0, would give

$$\frac{3{,}616{,}800 \times 3.0 \times 62.5}{2000} — 339{,}075 \text{ tons of ore available.}$$

339,075 × $7.73 — $2,621,050, total values.

EXAMPLE 4.

Triangular Grouping Possibly the simplest and most widely used method of combining irregularly spaced placer samples is by grouping into triangles. The area under consideration is subdivided into triangular blocks, with a drill hole or sample at each corner. An average assay for each block is computed by combining the three samples in proportion to their foot-assay products, and the volume-assay products of the individual triangular prismoids are then combined to give a final average assay value for the whole orebody.

The degree of irregularity of sample spacing, the uniformity of individual assays or the degree to which they are erratic, and the configuration of the body being sampled are the important factors to be considered in adapting this method to the sampling job. Particular care must be taken to avoid placing undue emphasis on any hole by centering too many triangles about it.

Fig. 8 is the assay map of a hypothetical placer deposit with samples grouped into triangles. A record of the computations is most readily kept by numbering each triangle and tabulating the data upon a form sheet, as on page 60.

It will be noted from the following calculations that the volume-assay product for each prismoid is, in reality, the value of the ore in the prismoid. By a tabulation of the results as given on the form sheet a direct allocation of values into individual prismoids is obtained in the process of arriving at the total value for the area and the average assay of the area.

The area of each triangle has been computed by two methods and the average result used to figure the volume of the prismoid. The first figure listed under triangle area is the product of the scaled base by one-half the scaled altitude; the second figure is the mean of two or more planimeter readings; the final figure is the average of the first two. It is obviously not necessary to use both these methods to compute the triangle

Δ No.	Sample Data		Foot-Assay Product	Average Assay	Average Thickness	Area of Triangle	Vol. of Prismoid Cu. Ft.	Vol. of Prismoid Cu. Yd.	Total Values in Prismoid
	20'@33¢		660						
1	15	38	570	1870/51 =	51/3 =	21534 Bx½A			
	16	40	640			21600 Pl.			
	51		1870	36.67¢	17.00 ft.	21570 Av.	366690	13581	$4980.20
	20'@33¢		660						
2	18	45	810	2040/53 =	53/3 =	24990			
	15	38	570			25000			
	53		2040	38.49¢	17.67 ft.	25000	441750	16361	6297.30
	18'@45¢		810						
3.	14	30	420	1800/47 =	47/3 =	35680			
	15	38	570			35900			
	47		1800	38.30¢	15.67 ft.	35790	560830	20771	7955.30
	16'@40¢		640						
4	15	38	570	1990/44 =	44/3 =	32175			
	13	60	780			32300			
	44		1990	45.23¢	14.67 ft.	32240	472960	17517	7922.90
	15'@38¢		570						
5	12	45	540	1890/40 =	40/3 =	32660			
	13	60	780			32900			
	40		1890	47.25¢	13.33 ft.	32780	436957	16184	7646.90
	15'@38¢		570						
6	14	30	420	1530/41 =	41/3 =	22400			
	12	45	540			22300			
	41		1530	37.32¢	13.67 ft.	22350	305525	11316	4223.10
	13'@60¢		780						
7	12	45	540	1830/42 =	42/3 =	30530			
	17	30	510			30600			
	42		1830	43.57¢	14.00 ft.	30560	427840	15846	6904.10
	12'@45¢		540						
8	19	50	950	2000/48 =	48/3 =	28512			
	17	30	510			28700			
	48		2000	41.67¢	16.00 ft.	28600	457600	16948	7062.30
	12'@45¢		540						
9	14	30	420	1910/45 =	45/3 =	28512			
	19	50	950			28700			
	45		1910	42.44¢	15.00 ft.	28600	429000	15889	6743.30
						Totals for Area =		144413	59735.30

$59,735.30 ÷ 144,413 cu. yd. = 41.36¢, average assay value per cubic yard.

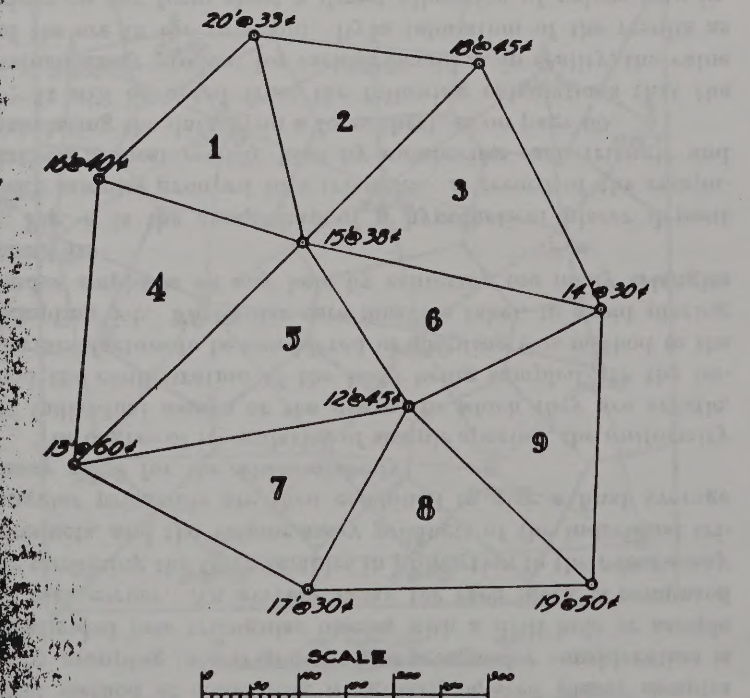


Fig. 8. Triangle grouping of irregularly spaced samples—arbitrary choice of triangles.

areas. They have been used here merely to show that comparable results may be obtained by the two types of mensuration; the final choice rests with the person who is performing the calculations. For extreme accuracy, the base and altitude of each triangle may be computed from totals of latitude and departure, or the area may be found by the double-meridian-distance method. No attempt has been made in these computations to carry the accuracy beyond the fourth figure; the field data are not sufficiently accurate to warrant such procedure.

EXAMPLE 5.

Configuration of Orebody When the orebody being sampled is extremely variable in thickness, it is evident that in grouping assays special precautions should be taken in choice of triangles to obtain accurate figures for value and tonnage. Probably the simplest way to take care of variable thickness is to contour the thickness of the orebody. This will make the irregularities apparent and be a guide to the engineer in grouping the area into triangles.

Fig. 9 represents a plan view of a portion of a placer deposit which has been churn-drilled at irregular intervals. The values given for each hole represent feet of gold-bearing gravel and value of the gravel in cents per cubic yard. For the sake of simplifying the problem, the gravel-bearing bed is assumed to be flat-lying, so that the dip or variations in dip need not be considered.

The first step is to sketch the thickness contours on the drill-hole plan. This is done by interpolating the thickness between each two adjacent drill holes and then connecting points of equal thickness. The resulting contours show that the top of the orebody, assuming the bottom to be flat, has an appearance similar to ordinary surface topography. Triangles are then laid out between holes to include, as nearly as possible, a uniform slope throughout each triangle.

The mathematical calculation of assay averages is no different in this method from the calculation in that just outlined. Greater accuracy, however, should be attained, because a complete picture of the orebody is made available and the engineer can adapt the method to the orebody variations to a much greater extent.

In actual practice it is permissible to assume one surface (either top or bottom) of the orebody to be flat only when it is a plane surface or nearly so or when both surfaces vary symmetrically from the median. The dip, when regular, can be

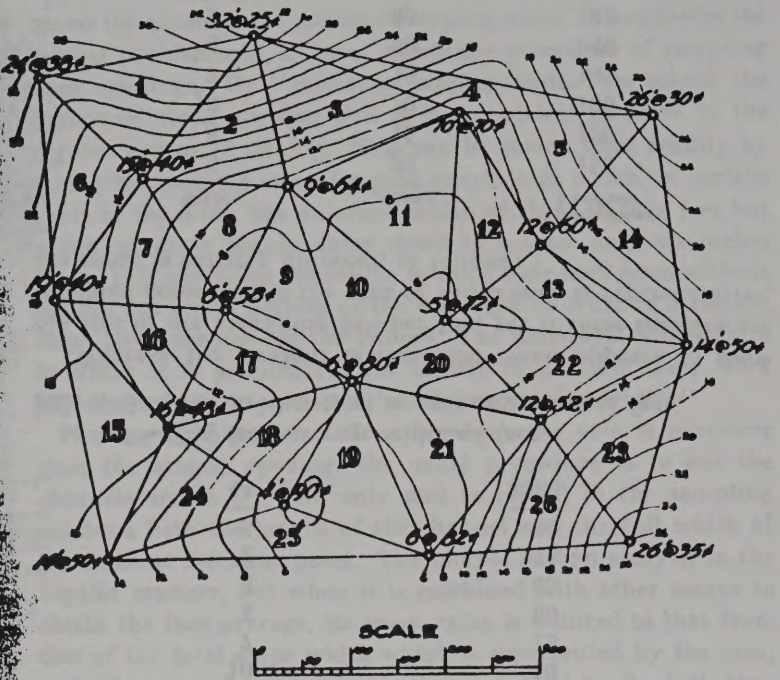


Fig. 9. Triangle grouping of irregularly spaced samples—based on thickness contours.

omitted from the contour map and from the immediate calculations and be applied as a general factor in computing the prismoid volumes. When both top and bottom contours (use colored inks to distinguish) are required to show the configuration of the orebody, the selection of triangles becomes doubly complicated because the samples must be grouped, if possible, so as to retain plane surfaces on both sides of the prismoids.

LINEAR GROUPS OF SAMPLES

EXAMPLE 6.

Arithmetic vs. Weighted Average Computing the arithmetical mean or straight unweighted average of a linear group of sample assays to obtain a composite value is a common mistake among the uninitiated, and has frequently resulted in serious error, particularly when rich or erratic ores are dealt with. This may be illustrated from the hypothetical group of assays of an 80-ft. portion of a gold quartz vein sampled at regular intervals of 10 feet by cross channels normal to the dip of the vein as shown in Fig. 10-A:

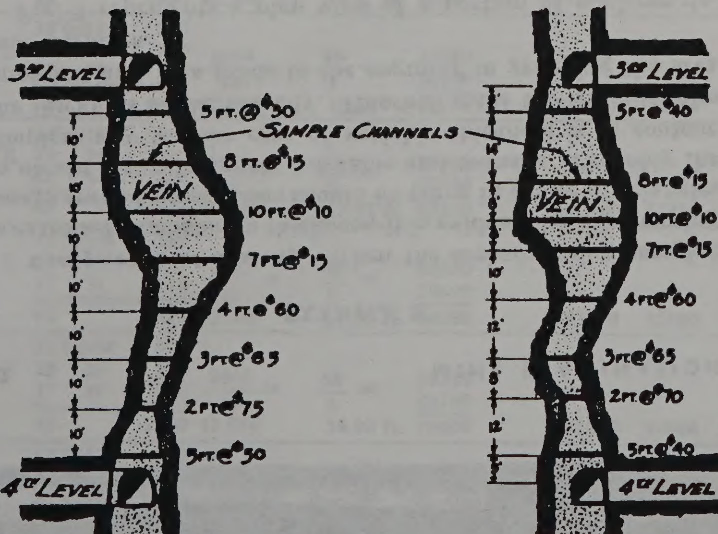


Fig. 10. Linear groups of channel samples.

Width	Assay
5 feet	$30 per ton
8	15
10	10
7	15
4	60
3	65
2	75
5	50
44 feet	$320

44 ÷ 8 — 5½ feet, average width of face.
$320 ÷ 8 — $40, arithmetical mean assay value of face.

Whereas the straight arithmetical mean appears to show 5½ feet of ore averaging $40 per ton, yet if these samples are weighted according to the area of which each is representative, the result is entirely different, as follows:

Width	Assay	Assay-Foot Product
5	30	150
8	15	120
10	10	100
7	15	105
4	60	240
3	65	195
2	75	150
5	50	250
44 feet		1310 (ft. $)

1,310 ÷ 44 — $29.77, average assay value of ore face, as contrasted with the higher figure of $40 per ton which was obtained by a straight arithmetical average of assay figures.

44 ÷ 8 — 5½ feet, average width of face.
80 × 5½ — 440 sq. ft., face area of vein.

EXAMPLE 7.

Distance of Influence A similar set of assays but with channels spaced at uneven measured distances is shown in Fig. 10-B. It is necessary, in this instance, to make some assumption in regard to extension of values in the vein beyond the end channels. This extension, as indicated, is taken as five feet, although from the given channel spacing, seven feet would be a reasonable assumption beyond the top channel and six feet below the bottom one. An extension of values for five feet assumes that the next channel beyond those shown on the sketch is 10 feet away.

This group of irregularly spaced samples would be averaged as follows:

Sample Width	Distance between Samples	Distance of Influence of Each Sample	Area of Influence of Each Sample	Assay of Sample	Area-Assay Product
	10 ft.				
5 ft.		12 ft.	60 sq. ft.	$40	2400
	14				
8		11	88	15	1320
	8				
10		7	70	10	700
	6				
7		8	56	15	840
	10				
4		11	44	60	2640
	12				
3		10	30	65	1950
	8				
2		10	20	70	1400
	12				
5		11	55	40	2200
	10		423 (sq. ft.)		13450 (sq. ft. $)

13,450 ÷ 423 = $31.80, average assay value of ore face.

Each assay, it will be noted, is multiplied by its length and by one-half the distance to the channels on each side, thus weighting each value by the area attributable to that value.

EXAMPLE 8.

Minimum Stoping Width In working narrow veins or thin seams the question of "minimum stoping width" complicates the mining problem and, in turn, affects the procedure of sampling such openings. By "minimum stoping width" is meant the narrowest stope opening that it is economical to drive in the regular course of mining. This can be shown most readily by taking the extreme case of a gold quartz vein which, in certain parts of the mine, has a normal width of three to four feet but which, at other points, thins down to a seam only six inches thick. It is obviously impossible to stope a six-inch seam without cutting out also a portion of the wall rock in order to make the stope large enough for the miners. The narrowest stope width in which it is possible for the miners to perform their work expeditiously is termed the "minimum stoping width."

In the sampling of a stope in which the vein is narrower than the stoped opening, the usual procedure is to cut the channels across the vein only and to record in the sampling notebook both the length of the channel and the full width of the stope at the same point. The sample is then assayed in the regular manner, but when it is combined with other assays to obtain the face average, its assay value is reduced to that fraction of the total stope width which is represented by the vein, and this reduced assay value is then weighted by the full stope width. In other words, the channel is assumed to extend the full stope width, and the assay of the vein itself is combined with a zero assay for the additional rock width to produce a discounted or diluted assay value for the full width.

A stope three feet wide is mined along a quartz vein. To obtain an estimate of values, the vein is cross-channelled at 10 ft. intervals. In one place the vein is only one foot wide. The assay of this particular sample is $90 per ton. In combining this sample with the others to calculate a face average, the value of $90 per ton for one foot of channel length must be averaged with that of two feet of barren rock because the stope

is three feet wide. The assay of $90 for one foot would, therefore, be equivalent to one of $30 for three feet and would be so used. When there are evidences that the mineralizing solutions have so penetrated the country rock of one or both walls as to give irregular and minor assay values to this rock, the use of a sample channel the full width of the expected stope will give a more accurate total result.

To illustrate further, take the sampling data listed above for Fig. 10-A and assume that in working this deposit the minimum stoping width is five feet. The average assay value of the face would then be computed as follows:

Stope Width	Vein Width	Assay	Assay-Foot Product
5	5	30	150
8	8	15	120
10	10	10	100
7	7	15	105
5	4	60	240
5	3	65	195
5	2	75	150
5	5	50	250
50			1310

1,310 ÷ 50 = $26.20, average assay value of the stope face.
50 ÷ 8 = 6.25 feet, average width of stope face.
80 × 6.25 = 500 sq. ft., area of stope face.

It will be noted that in this instance the assays of all vein widths of less than the minimum stope width of five feet are diluted with zero assays to apportion their values over the required stoping face. This is most readily done by taking the assay-foot products in the regular manner and then dividing the total of the assay-foot products by the total of the face widths at the points sampled instead of by the total of the vein widths as in the previous problem.

In estimating ore values, it is sometimes advisable to sample only the orebody and apply to the values so found the cost of mining necessary to extract these values. In this way the rock dilution attendant upon thin-vein mining is applied as additional cost rather than as discounted assays. The final net result is the same in both cases. This procedure is especially applicable if sorting can be practiced underground and the waste material stowed in the openings as fill, so that only the ore is hoisted and treated.

BLOCKS OF ORE

EXAMPLE 9.

Combining Average Face Assays In order to demonstrate the combining of face assays so that the average assay of a block of ore may be obtained, assume a 30 × 40 ft. block of ore in a gold quartz vein which has been sampled on two sides at regular 10-ft. intervals by cross channels as shown in Fig. 11. The calculations are as follows:

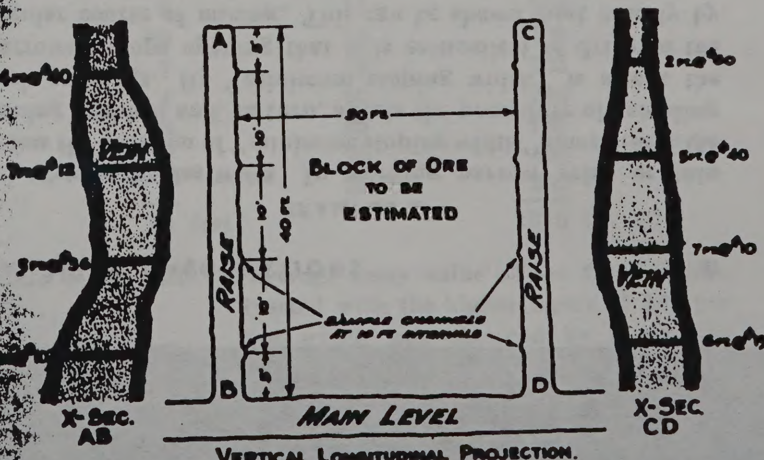


Fig. 11. Block of ore defined by sampled faces.

FACE AB			FACE CD		
Sample Width	Assay Value	Foot-Assay Product	Sample Width	Assay Value	Foot-Assay Product
4 ft. @	$40 —	160	2 ft. @	$60 —	120
7 @	12 —	84	5 @	40 —	200
5 @	36 —	180	7 @	10 —	70
8 @	10 —	80	6 @	15 —	90
24 ft.		504	20 ft.		480

FACE AB	FACE CD
504 ÷ 24 — $21, average assay value of face.	480 ÷ 20 — $24, average assay value of face.
24 ÷ 4 — 6.0, average width of face.	20 ÷ 4 — 5.0, average width of face.
40 × 6.0 = 240 sq. ft., area of face.	40 × 5.0 — 200 sq. ft., area of face.

$21 × 240 sq ft. — 5040, area-assay product of face AB.
$24 × 200 sq. ft. — 4800, area-assay product of face CD.
9840 ÷ 440 — $22.36, average assay value of entire block.

Assuming the specific gravity of the ore to be 3.0, the tonnage and total value of the ore in the block are as follows:

$$30 \times 40 \times \left(\frac{6.0 + 5.0}{2}\right) \text{ — volume of block — 6600 cu. ft.}$$

$$\frac{6600 \times 3 \times 62.5}{2000} = 618.75\text{, short tons in block.}$$

618.75 × $22.36 = $13,835.25, total value of ore in block.

EXAMPLE 10.

Estimating Average Assay Value of Blocks of Ore When estimating the average value of a block of ore that is exposed on two adjacent sides (two sides being necessarily adjacent when a block is exposed on three or four sides), the engineer encounters

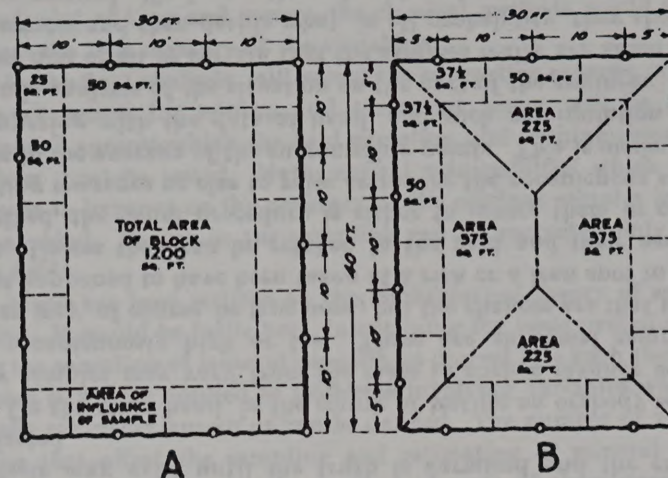


Fig. 12. Weighting of uniformly spaced samples.

a problem in weighting which belongs among the debatable refinements mentioned previously. To demonstrate this situation, take an example such as shown in Fig. 12, in which a rectangular block of ore 30 feet by 40 feet is bounded on all four sides by mine openings, with ore values visible on each exposed face. In order not to complicate the problem further, assume the block to be a portion of a uniform and relatively thin vein, so that the drifts and raises expose the whole thickness of the vein on each side. The problem is to sample this block by cross channels at 10-ft. intervals and to estimate the average assay value of each face, the tonnage available, the average assay of the entire block, and the total values available. As shown in the longitudinal section, the channels may be laid out in two ways — with the end channels of each face at the corners of the block or with the end channels of each face located at a half-interval (5 feet) from the corners so that there are no common channels.

Sample Spacing Either of these layouts will demonstrate the point in question — namely, how to weight the individual samples and the average assays of the various faces when combining them to compute an average assay for the entire block.

The average assay of each face can be computed for either of these layouts by the foot-assay-product method already explained, with one modification: the assay value of the corner channels (Fig. 12-A) must be given only one-half the weight of the intermediate channel assays because the area of influence attributable to these corner samples is only one-half that of the others. This may be readily seen by reference to the sketch. Also, if strict adherence is paid to the areas of influence, the end samples on each face in the layout shown in sketch B should be weighted only three-fourths as heavily as the intermediate samples; in fairly uniform orebodies, however, this is rather an unnecessary and impractical refinement, one of those controversial issues that continually crop out when sampling problems are under discussion. If the orebody is extremely rich or erratic, the channels will be spaced at closer intervals, and the end values will drop in relative importance because of the larger number of intermediate values with which they are combined.

Face Influence According to the methods already outlined, the average assay of each face is weighted according to its respective face area when combining to obtain a single average for the block. This, though a practical weighting, is not mathematically exact, as can be seen by referring to the dot-and-dash-lined (- . . . -) areas of sketch B, wherein, of a total of 1200 sq. ft. of area in the entire block, 375 sq. ft. are apparently attributable to each of the long sides, and 225 sq. ft. to each of the short faces. This gives a ratio of 3:5 for the ends to the sides instead of a 3:4 ratio based on the lengths of the sides and ends. Assuming uniform thickness of vein throughout, this difference would change the volume weights by 15 percent.

TREATMENT OF ABNORMAL ASSAYS — In the sampling of any orebody the purpose is to have the samples represent the ore at all points. An orebody necessarily fluctuates in value content from one point to another. It may have lean or barren spots; it may have rich spots. Sampling is intended to show these variations. Any cut-and-try procedure such as sampling, however, is liable to error; the samples may skip a lean or rich spot or they may, by chance, hit only the lean or only the rich spots. These eventualities give erroneous results and, since the results are seemingly consistent and the field data apparently normal, serious operating defects may exist until the truth is surmised and the error detected.

On the other hand, in the course of testing an orebody some few samples may vary from the mass in such a manner as to be conspicuously high or low. These are abnormal samples. They may, of course, be true ones; but the chances are that they just happened to have been taken at a rich or a lean spot in the ore. Unless they can be retaken in the field and their assays verified, the usual procedure is either to ignore them in computing averages or else to raise or reduce the conspicuous samples to the average of the surrounding points. This is obviously tampering with the data at hand, and such manipulation demands analysis of the situation on the part of the engineer. He must first of all be certain that the samples really are abnormal or erratic and then decide what, in all probability, they should have assayed. To be on the safe side, when only a few such values are involved, most engineers scale down the conspicuously high samples but seldom raise the low ones.

USE OF EXPERIENCE FACTORS — In most sampling situations, the estimation of tonnage and values can be handled adequately by the classical methods. These are the methods of sampling and weighting that have been described; they are fundamentally sound in principle, rela-

tively simple to apply, and well understood by engineers and geologists.

Exceptional situations may call-for special procedures in the combining of assays to give accurate results. These, generally, will relate to specific characteristics in the material being sampled, such as particle size and distribution, or loss by solubility of critical elements or contaminants. An example might be a placer gold deposit in which much of the gold occurs as fairly uniform, fine particles but with which are associated some amount of large particles and nuggets. The deposit is tested by drill holes. The larger particles, being much less in number in proportion to their value, have less chance of being included in a sample than the uniformly distributed small particles. Further, inclusion of a large particle, or two such particles, or a nugget in a sample will give higher than average to exceptionally high assays. These may appear to be erratic values. But if all the drill holes, by chance, missed large particles, the final result would be below the true value. In sampling and estimating a deposit of this type, some gauge is needed to bring the values into their true relationship, one with another. Analysis by grain count and probabilities[1] may offer a guide; so also may consideration of frequency distribution[2] of values among large groups of samples. More often, experience gained by operating companies through years of correlating daily field samples with final production will have developed suitable corrective measures to compensate for such troublesome values. The examiner will do well to seek out these experience factors whenever the situations that prompted their adoption are comparable to the conditions surrounding his problem. This is not to say that, carried to their ultimate, which often is impractical from the

[1] Swanson, C. O., "Probabilities in Estimating the Grade of Gold Deposits," C.I.M.M. Trans., Vol. 48, 1945, pp. 323-50.

[2] Truscott, S. J., "Mine Economics," Mining Pub., Ltd., London, 2nd Ed., 1947, Chap. 6.

standpoint of time and money, the classical methods would not reveal the true measure of the variables.

In the final analysis, mill recoveries or smelter returns, with due allowance for mining and treatment losses, will be the standard against which the field sampling and ensuing computations must be tested. Mathematical manipulation which attempts to improve on the accuracy of the original samples is to be avoided. Errors in the sampling can be corrected only in the field.

Much has been written on the mathematical theory of sampling. It would be futile here, in outlining the basic procedures for the sampling of mineral deposits, to digress into such theory which is widely applied to problems involving variables whose limits are either known or can be defined. The number of variables that affect the sampling and estimating of mineral deposits and the indeterminate nature of some of them prevent rigorous solution. The assumptions that must be made in regard to these variables, in large part at least, are not susceptible to rigid formulae, since they relate to the geologic nature of the problem.

THIS PAGE
INTENTIONALLY
BLANK

COMPUTING RESERVES OF MINERAL DEPOSITS: PRINCIPLES AND CONVENTIONAL METHODS

By Constance C. Popoff

* * * * * * * * * * * * Information Circular 8283

UNITED STATES DEPARTMENT OF THE INTERIOR
Stewart L. Udall, Secretary

BUREAU OF MINES
Walter R. Hibbard, Jr., Director

USBM IC [illegible]33
Computing Reserves
of Mineral Deposits

COMPUTING RESERVES OF MINERAL DEPOSITS: PRINCIPLES AND CONVENTIONAL METHODS

By Constantine C. Popoff

* * * * * * * * * * * information circular 8283

UNITED STATES DEPARTMENT OF THE INTERIOR
Stewart L. Udall, Secretary

BUREAU OF MINES
Walter R. Hibbard, Jr., Director

Reprinted by:
Bureau of Land Management
Phoenix Training Center

This publication has been cataloged as follows:

Popoff, Constantine C

Computing reserves of mineral deposits: principles and conventional methods. [Washington] U. S. Dept. of the Interior, Bureau of Mines [1966]

113 p. illus., tables. (U. S. Bureau of Mines. Information circular 8283)

Includes bibliography.

1. Mines and mineral resources. I. Title. (Series)

TN23.U71 no. 8283 622.06173

U. S. Dept. of the Int. Library

CONTENTS

CONTENTS--Continued

CONTENTS--Continued

ILLUSTRATIONS

ILLUSTRATIONS--Continued

TABLES

THIS PAGE
INTENTIONALLY
BLANK

COMPUTING RESERVES OF MINERAL DEPOSITS: PRINCIPLES AND CONVENTIONAL METHODS

by

Constantine C. Popoff[1]

ABSTRACT

This report reviews and analyzes, by a simple analytical and logical reasoning, the conventional methods of reserve computations of mineral deposits described in various domestic and foreign publications. It brings together, formulates, and evaluates the principles underlying interpretation of exploration data; and ties such principles to the proposed classification of methods. The material is discussed in sufficient detail to allow general application.

INTRODUCTION

Computation of reserves is recognized by the mineral industry as a distinct operation of increasing importance in the evaluation of mineral deposits in all stages of their development. Previously, valuation was based on facts, experience, and intuition; methods have inproved because our knowledge of mineral deposits, sampling, and mining techniques has increased.

Originally, computation methods followed practices of earth excavation and road construction, both standard surveying operations. Advances in earth sciences and engineering resulted in the modification of old and introduction of new methods.

The purpose of this investigation is to review some of the common methods and their modifications used in reserve computations of mineral deposits. The scope of this paper is limited to solid mineral deposits (that is, metal, nonmetallic, coal, and oil shale), because the background data required, procedure, and methods for water, oil, and gas are dissimilar. An attempt is made to systematize and standardize the methods and terminology.

For convenience, the paper is divided into two parts. The first, "Principles", deals with assumptions and scientific principles underlying the

[1] Former mining engineer, Bureau of Mines, Area VII, Seattle Office of Mineral Resources, Seattle, Wash.

Work on manuscript completed February 1965.

use of various methods, and provides a general discussion of the elements of computations, procedure, and errors of interpretation. The second part, "Conventional Methods", covers the following methods and their modifications: average factors and area (analogous and geologic blocks), mining blocks, cross sections (standard, linear, and isolines), triangular and polygonal prisms, and combinations of these.

The text often refers directly to ore deposits, because the problem of computing their reserves is generally more complicated due to diversity of form and size of mineral bodies and irregular distribution of values. The same methods are used for coal and nonmetallic deposits.

Statistical analysis is a valuable tool of research for all the methods of computations. Application of various methods of statistical analysis to sampling and exploration data is under continuing investigation by the Bureau of Mines. These methods and the use of computers for reserve computations are discussed by the Bureau and other scientists in several recent publications. They are beyond the scope and intent of this paper.

Reference is made to "exploration" workings for brevity; however, the text applies to mineral deposits in all stages of exploration, development, and exploitation. In the latter case, computations are particularly critical from the standpoint of economics but, once accepted, are usually subsidiary and routine operations.

ACKNOWLEDGMENTS

Grateful acknowledgment is made to J. A. Patterson, assistant chief of Ore reserve Branch, U.S. Atomic Energy Commission, Grand Junction, Colo., for the opportunity to study unpublished material on reserve computations.

PART 1. - PRINCIPLES

General

Significance of Computations

The purpose of reserve computations of a mineral body is to determine the quantity, the quality, and the amenability to commercial exploitation of raw material (ore, rock, coal, etc.). Computations are made during all stages of the life of a mining enterprise from discovery to robbing pillars and closing. They are the most responsible and irreplaceable tasks in the valuation of a mineral deposit. Efficiency in extraction and productiveness is impossible without accurate reserve computations.

Reserves are computed to determine the extent of exploration and development; distribution of values; annual output; probable and possible productive life of the mine; method of extraction and plant design; improvements in extraction, treatment, and processing; and requirements for capital, equipment, labor, power, and materials. Such computations are used to assist development planning; to determine production costs, efficiency of operations, and mining

losses; for quality control; for financing mining ventures; for sale, purchase, and consolidation of companies; to determine the production cost per unit of a marketable product; for accounting purposes such as depletion and depreciation; and in some States for tax purposes.

Requirements

No computations are justified unless called for and used; they should be made when required. The ideal method should be simple, rapid, reliable, consistent with the character of the mineral body and available data, and suitable for rapid checking. Computations are expected to be inexpensive when compared with the cost of exploration and development, and therefore, more complex methods are sometimes justifiable, particularly when labor-saving devices (calculators and computers) are available. In selecting a method, the peculiarities and conveniences of automation should be considered, as well as the magnitude and accuracy required.

The method should be selected carefully, procedures worked out in detail, and computations made accurately. Formulas should be simple. Properly selected procedures will facilitate the process of computations and provide the same degree of accuracy as more complicated methods.

Objective treatment of factual data is considered by many earth scientists the most important requirement. Harding, for example, states that his studies and formulas were provided by a desire to find "a method of calculating which eliminates all factors of test and judgment and rests on pure mathematics, ...a method which can be handled almost entirely by a calculating machine" (13).[2]

Computations should also meet the purpose of the valuation and, when appropriate, illustrate the distribution of variables.

The reliability of reserve computations depends chiefly on the accuracy and completeness of our knowledge of the mineral deposit. It also depends on assumptions accepted for interpreting the variables, on boundaries of mineral bodies, on accuracy of averages, and on mathematical formulas. Requirements for the quantity and the density of observations for a certain category of resources depends primarily on the size and type of the mineral deposit.

During the last several decades the accuracy of computing reserves has gradually improved. This was made possible by outstanding advances in the field of economic geology; increased specialization; improvements in exploration, sampling, mining, and valuation; better interpretation of field information; use of statistical analysis; and more efficient management.

The growing use of data-processing machines has made it possible to record large amounts of exploration data in the form of punch cards, punched tape, magnetic disk, or magnetic tape. The computers permit application of

---

[2] Underlined numbers in parentheses refer to items in bibliography at the end of this report.

two or more conventional methods and produce improved accuracy, increased speed, and labor and cost savings in reserve computations. The techniques and advantages of the use of computers are discussed in several recent publications (12, 21, 23-25, 38).

Criteria for Method Selection

In general, selecting a method for reserve computations depends upon the geology of the mineral deposit, exploration method, availability and reliability of factual data, purpose of computations, and the required degree of accuracy.

If computations are preliminary or are required immediately, simple methods, which do not demand construction of special maps, are selected. If computations are for mine design, the method selected depends on the contemplated mining system. The cutoff grade, recovery, dilution, efficiency of equipment and labor, and cost per unit of output vary with the system of extraction. A simple method may be adequate for open pit operations when selective extraction of waste or weakly mineralized rock is excluded. Computations of reserves for a bedded deposit is less complex than for high-grade, small volume, stock-type deposits with irregularly distributed values.

Exploration, whether random, by grid, or by cross-section lines, may also influence method selection. It is often desirable during exploration to use a method permitting step-by-step addition of reserves to previous figures instead of periodic recomputations.

The nature of the various methods should be carefully considered. Simple methods are preferred, but more complicated ones may be justified. Both extremes, oversimplification leading to complete disregard of the geologic nature of the deposit and overcomplication leading to unwarranted precision, expense, and even impracticability, should be avoided. The question of maximal use of all factual data collected in the process of exploration is an important consideration. Poor planning and overexploration results in excessive data not necessary for the accepted accuracy of computations.

Computing Reserves Procedure

Analysis of Exploration Data

Reserve computations of a mineral deposit is a technical task, consisting of several operations. The importance of following a definite procedure, properly selected for a certain deposit, cannot be overemphasized (41). The operations in order of their usual execution are geologic appraisal, exploration and sampling methods appraisal, exploration data appraisal, delineation of the mineral body, and selection of an appropriate method for computations.

The importance of the knowledge of the geology of the deposit for the understanding of the size, shape, and grade distribution, and for interpretation of exploration data has been emphasized by many scientists (29, 34). Geologic appraisal includes obtaining, checking, and presenting exploration

data in the form of graphs, tables, maps and sections of appropriate scale, and assuming a working hypothesis on the origin of mineralization. The exploration method; that is, the kind and density of workings and sampling, is studied to determine the adequacy and accuracy of the data from the standpoint of geology, geometric configuration of the mineral bodies, distribution pattern of variables, errors, and category of reserves. Such an appraisal is often supplemented by statistical analysis and by comparison with other deposits similar in type and form.

The analysis of exploration data, often the most neglected step in valuation, is accomplished by defining inside and outside parameters of economically minable portions of the mineral body; by determining the precision of measurements and analyses; and by determining whether the amount of exploration of various portions of the mineral body meets the requirements for computing reserves of a certain category.

Procedure

For reserve computations the mineral body is first delineated and then subdivided by several methods into segments or blocks of various degrees of reliability.

The usual procedure for volume computations is to substitute graphically the irregular shape of the mineral body by an imaginary and auxiliary one with base surface lying in the plane of a plan or longitudinal section; the other surface, irregular in form, shows distribution of thicknesses (fig. 1). This auxiliary body is then replaced by one or several simple solid figures, volumes of which can be computed by geometric formulas.

Division of the mineral body into blocks is done according to a selected method, so that each block can be directly related to one or a suite of factual exploration values.

The reserves of the entire body are computed by determining areas and volumes for each block, converting block volumes to tonnages of raw mineral material, determining average grades and tonnages of valuable components, and finally, tabulating the result of blocks of the same category and, if possible, assessing the reliability of computations.

Main Elements

Reserve computations require a knowledge of the dimensional and qualitative features of the mineral body. This knowledge is gained directly by observations (measurements, chemical analyses, and tests) and indirectly by assumptions, interpretations, and computations. All values of the basic block elements, thickness, length, breadth, weight factor, and grade, whether they are single observations or computed averages, may be presented on maps by numbers pinpointed for a definite location, or as a line with the numerical length plotted to scale.

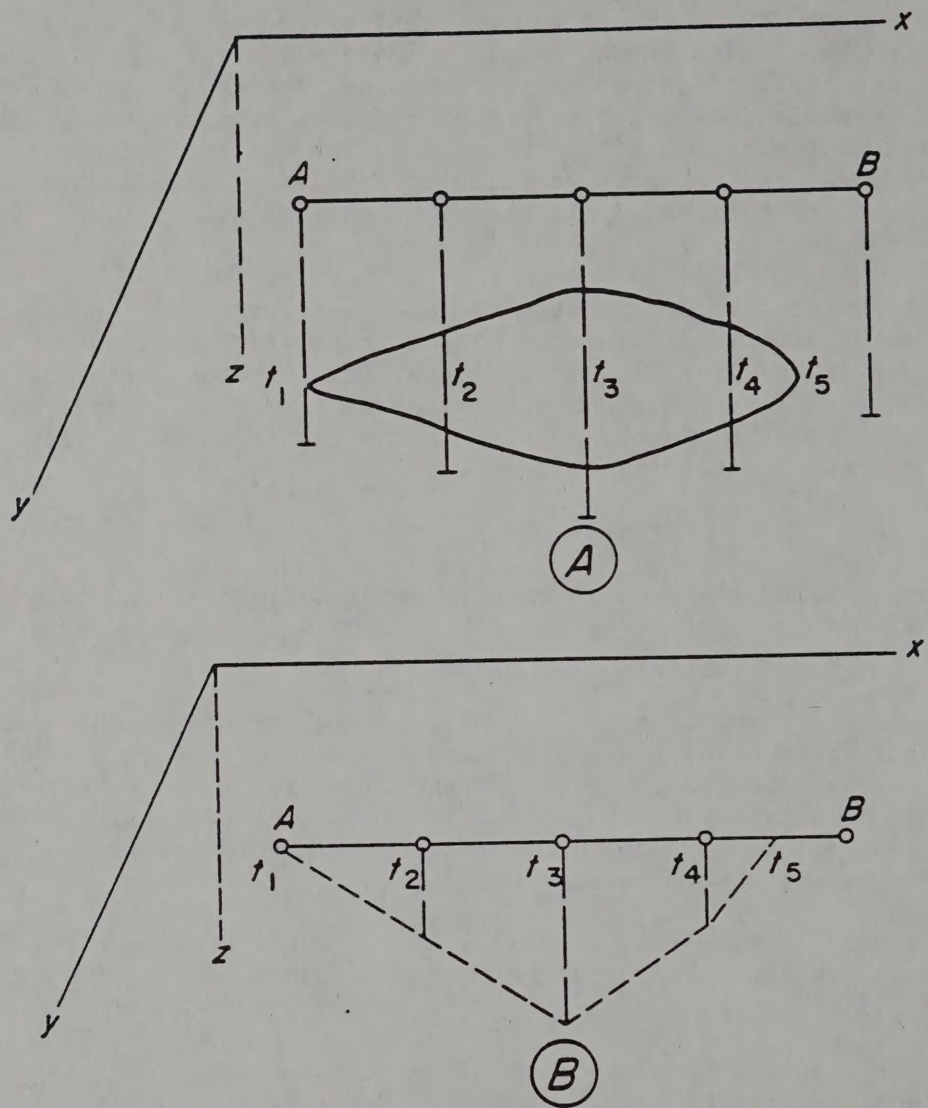


FIGURE 1.- Transforming a True Mineral Body Into an Imaginary Auxiliary One. *A*, Vertical section of true body; *B*, vertical section of distorted auxiliary body.

The system selected for measuring linear distances, areas, volumes, and weights should be followed throughout. Units of measure and weight and conversion factors for English and metric systems are given in appendix A. When selection is possible, the metric system is preferable; it saves time and reduces the chance of error.

The formulas for all methods are based on computing solids with their bases constructed in the plane; vertical thickness is used for horizontal plan; horizontal thickness for vertical or longitudinal sections; and true thickness for incline longitudinal sections drawn in the plane of the dip of the mineral body. The relationships between the true, horizontal, and vertical thicknesses are:

$$t_{tr} = t_h \sin\beta = t_v \cos\beta \qquad (1)$$

where β is the true dip of the body (fig. 2A).

In reserve computations, the true strike and the true dip of the deposit, or its portion under consideration, are determined first of all. Corrections for dip taken in a direction not perpendicular to the strike are made from specially prepared tables or by a protractor described in field geology textbooks (11, 33).

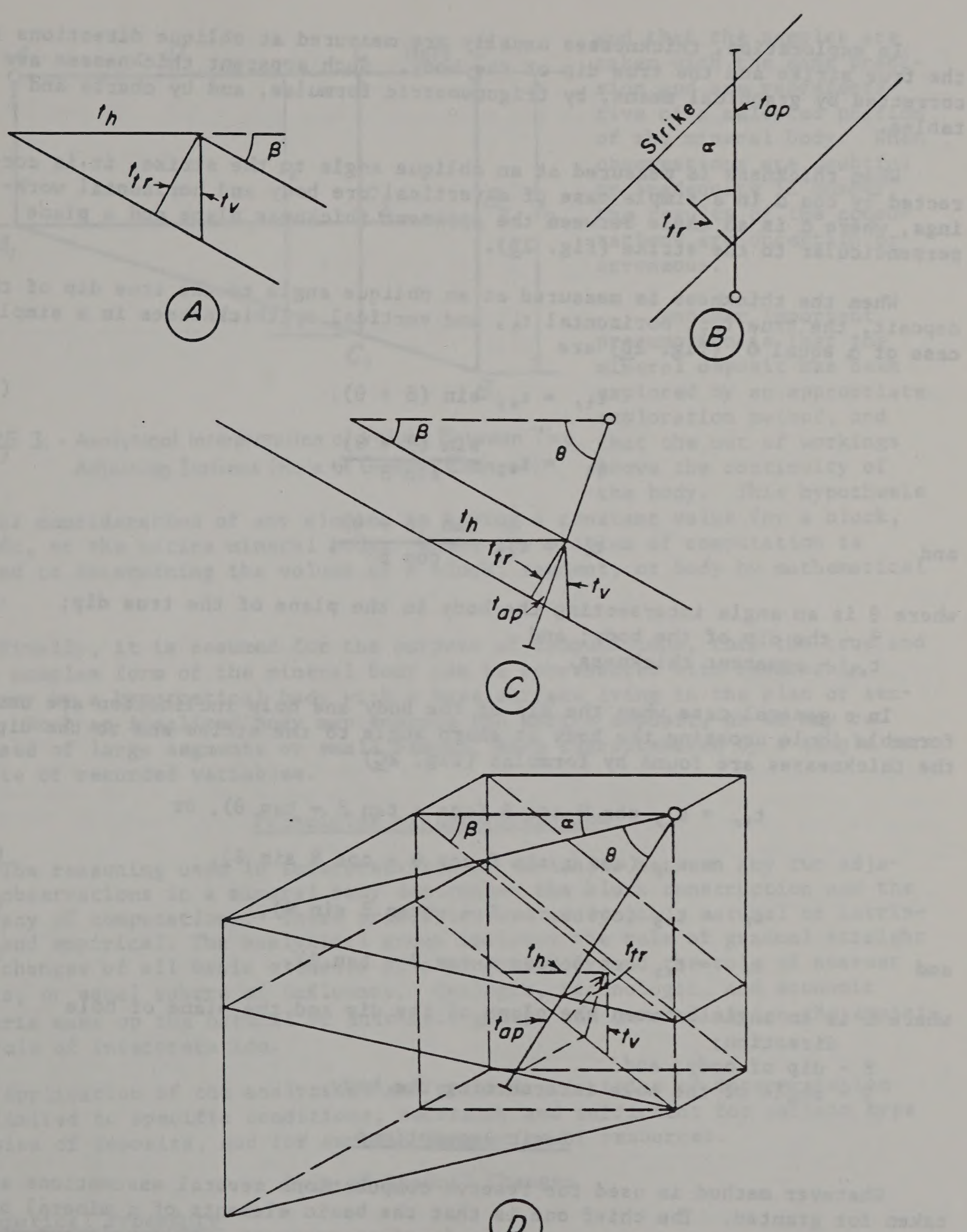


FIGURE 2. - True, Horizontal and Vertical Thicknesses—Analytical Relationship. *A*, $t_{tr} = t_h \sin \beta = t_v \cos \beta$; *B*, strike correction - $t_{tr} = t_{ap} \cos \alpha$ ($\beta = 90°$ $\theta = 0$); *C*, dip correction ($\alpha = 0°$); *D*, general case - block diagram.

In exploration, thicknesses usually are measured at oblique directions to the true strike and the true dip of the body. Such apparent thicknesses are corrected by graphical means, by trigonometric formulas, and by charts and tables.

When thickness is measured at an oblique angle to the strike, it is corrected by cos α in a simple case of a vertical ore body and horizontal workings, where α is an angle between the apparent thickness plane and a plane perpendicular to the strike (fig. 2B).

When the thickness is measured at an oblique angle to the true dip of the deposit, the true t_{tr}, horizontal t_h, and vertical t_v thicknesses in a simple case of α equal 0° (fig. 2B) are

$$t_{tr} = t_{ap} \sin(\beta + \theta), \tag{2}$$

$$t_h = t_{ap} \frac{\sin(\beta + \theta)}{\sin \beta}, \tag{3}$$

and

$$t_v = t_{ap} \frac{\sin(\beta + \theta)}{\cos \beta}, \tag{4}$$

where θ is an angle intersecting the body in the plane of the true dip;
β - the dip of the body; and
t_{ap} - apparent thickness.

In a general case when the dip of the body and hole inclination are unconformable (hole crossing the body at sharp angle to the strike and to the dip), the thicknesses are found by formulas (fig. 2C)

$$t_{tr} = t_{ap} \cos\beta \cos\theta (\cos\alpha \tan\beta + \tan\theta), \text{ or}$$

$$= t_{ap} (\cos\alpha \sin\beta \cos\theta + \cos\beta \sin\theta), \tag{5}$$

$$t_h = t_{ap} (\cos\alpha \cos\theta + \cot\beta \sin\theta), \tag{6}$$

and

$$t_v = t_{ap} \cos\theta (\cos\alpha \tan\beta + \tan\theta), \tag{7}$$

where α is an angle between the plane of the dip and the plane of hole direction;
β - dip of body; and
θ - angle of the hole intersecting the body.

Basic Assumptions

Whatever method is used for reserve computations several assumptions are taken for granted. The chief one is that the basic elements of a mineral body observed or established at any station (surface exposure, drill hole, or underground workings) change or extend to the adjoining area according to an appropriate principle of interpretation of exploration data. It is assumed also that observations are made in conformity with the nature of a given deposit

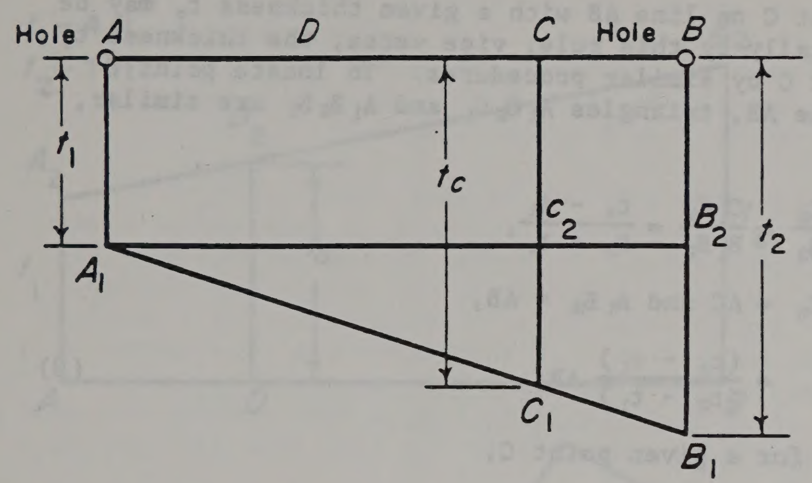


FIGURE 3. - Analytical Interpretation of Values Between Two Adjoining Stations (Rule of Gradual Changes).

and that the samples are taken with the same precision and are representative of a selected portion of the mineral body. When observations are doubtful or inadequate in number, the results of the computations are uncertain or erroneous.

Another important presumption is that the mineral deposit has been explored by an appropriate exploration method, and that the net of workings prove the continuity of the body. This hypothesis permits consideration of any element as having a constant value for a block, segment, or the entire mineral body. Thus, the problem of computation is reduced to determining the volume of a block, segment, or body by mathematical means.

Finally, it is assumed for the purpose of computations, that the true and often complex form of the mineral body can be represented with reasonable accuracy by a hypothetical body with a base surface lying in the plan or section. Such an idealized body may embrace the entire deposit, or it may be composed of large segments or small blocks, each characterized by a single or a suite of recorded variables.

Principles of Interpretation

The reasoning used in interpretation of variables between any two adjacent observations in a mineral body determines the block construction and the accuracy of computations. These principles are analytical, natural or intrinsic, and empirical. The analytical group includes the rule of gradual straight line changes of all basic elements of a mineral body and the rule of nearest points, or equal sphere of influence. Geologic, technologic, and economic criteria make up the natural or intrinsic group, and generalization the empirical rule of interpretation.

Application of the analytical and natural principles of interpretation are limited to specific conditions, necessary and sufficient for certain type and size of deposits, and for certain categories of resources.

Rule of Gradual Changes

Mathematical Procedure

According to the rule of gradual changes or law of linear function, all elements of a mineral body that can be expressed numerically change gradually and continuously along a straight line connecting two adjoining stations (fig. 3). Let us consider two adjoining stations or holes A and B with thicknesses

t_1 and t_2. Location of a point C on line AB with a given thickness t_c may be found analytically and graphically by this rule; vice versa, the thickness t_c may be found for a given point C by similar procedures. To locate point C with given thickness t_c on line AB, triangles $A_1C_2C_1$ and $A_1B_2B_1$ are similar, thus

$$\frac{A_1C_2}{A_1B_2} = \frac{C_1C_2}{B_1B_2} = \frac{t_c - t_1}{t_2 - t_1},$$

$$A_1C_2 = AC \text{ and } A_1B_2 = AB,$$

$$AC = \frac{(t_c - t_1)}{(t_2 - t_1)} AB; \tag{8}$$

and to determine thickness t_c for a given point C,

$$\frac{C_1C_2}{B_1B_2} = \frac{A_1C_2}{A_1B_2} \text{ or } \frac{(t_c - t_1)}{(t_2 - t_1)} = \frac{AC}{AB},$$

$$t_c - t_1 = \frac{AC}{AB}(t_2 - t_1),$$

and

$$t_c = \frac{AC}{AB}(t_2 - t_1) + t_1. \tag{9}$$

In surveying, equations (8) and (9) are known as formulas of simple interpretation.

The rule of gradual changes can be applied to other parameters of a mineral body such as grade and weight factors, as well as to areas, linear reserves, volumes, and tonnages. It may be used also in delineating the commercial portion of the deposit and to determine a given value at an unknown point on the extension of a line beyond known stations. In practice, interpolation and extrapolation are done by graphic means.

Graphic Procedure

To determine by vectors point D with a given thickness t_d of 5 feet, on a line AB with t_1 equal to 4 feet at station A and t_2 equal to 7 feet at station B, raise a perpendicular from station B equal to $t_2 - t_d$, or 2 feet, and drop a perpendicular from station A equal to $t_d - t_1$, or 1 foot (fig. 4). Connect points A_2 and B_2 with a straight line; the intersection of lines A_2B_2 and AB is point D.

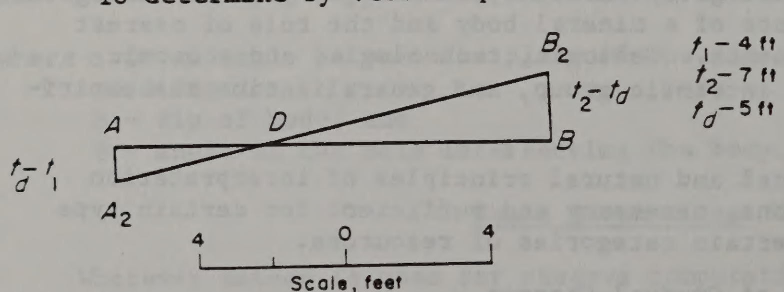


FIGURE 4. - Finding the Location of Point D With Thickness Five Feet by Means of Vectors (Rule of Gradual Changes).

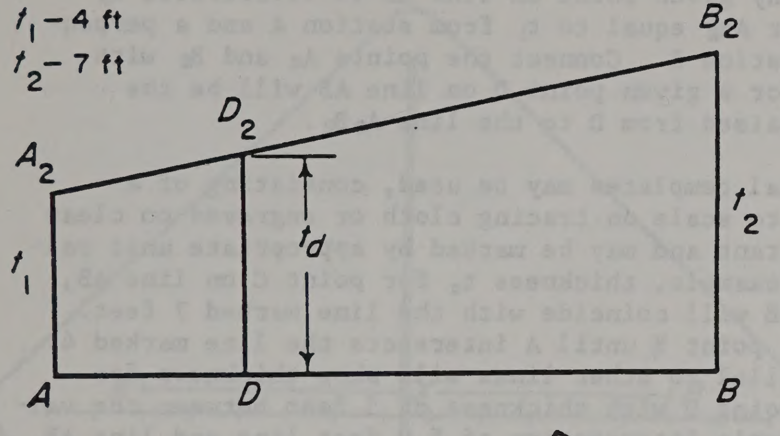


FIGURE 5. - Finding Thickness t_d for Point D (Rule of Gradual Changes).

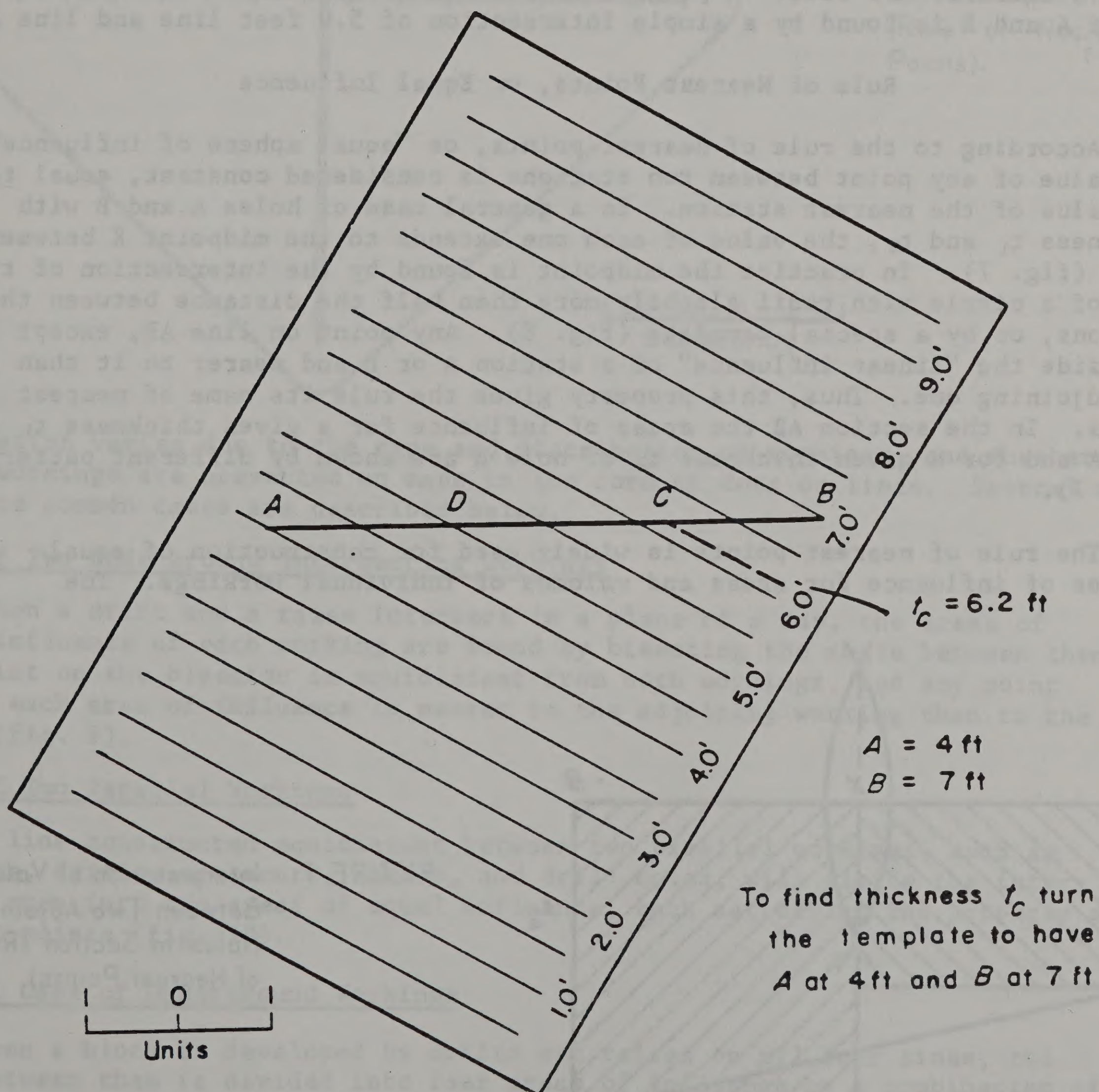


FIGURE 6. - Finding Thickness t_c for Point C and Finding Point D for Thickness t_d = 5 Feet by Means of a Special Template (Rule of Gradual Changes).

Finding thickness t_d for any given point on line AB is illustrated by figure 5. Raise a perpendicular AA_2 equal to t_1 from station A and a perpendicular BB_2 equal to t_2 from station B. Connect the points A_2 and B_2 with straight line. The thickness for a given point D on line AB will be the length of a perpendicular DD_2 raised from D to the line A_2B_2.

For the same purpose special templates may be used, consisting of a series of parallel lines drawn to scale on tracing cloth or engraved on clear plastic. The lines are equidistant and may be marked by appropriate unit values (fig. 6). To find, in our example, thickness t_c for point C on line AB, the template is placed so that B will coincide with the line marked 7 feet. Let us turn the template around point B until A intersects the line marked 4. A supplementary line drawn parallel to other lines will show thickness for point C equal to 6.2 feet. A point D with thickness of 5 feet between the values of A and B is found by a simple intersection of 5.0 feet line and line AB.

Rule of Nearest Points, or Equal Influence

According to the rule of nearest points, or "equal sphere of influence", the value of any point between two stations is considered constant, equal to the value of the nearest station. In a general case of holes A and B with thickness t_1 and t_2, the value of each one extends to the midpoint X between holes (fig. 7). In practice the midpoint is found by the intersection of two arcs of a circle with radii slightly more than half the distance between the stations, or by a special template (fig. 8). Any point on line AB, except X, is inside the "linear influence" of a station A or B and nearer to it than to the adjoining one. Thus, this property gives the rule its name of nearest points. In the section AB the areas of influence for a given thickness t_1 of hole A and for a given thickness t_2 of hole B are shown by different patterns (fig. 7).

The rule of nearest points is widely used for construction of equal spheres of influence for areas and volumes of individual workings. The

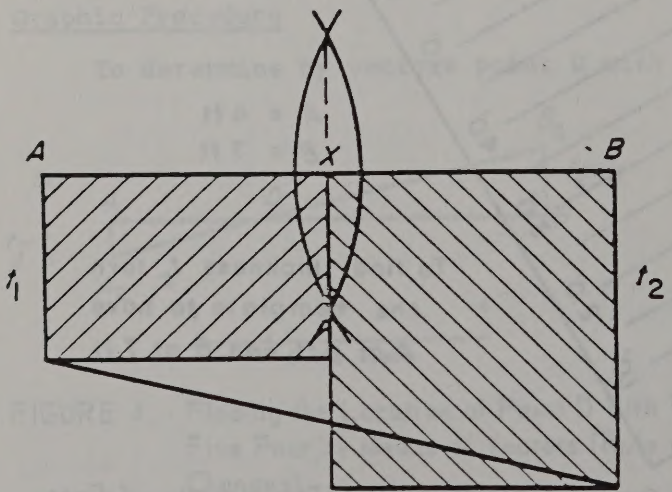


FIGURE 7. - Interpretation of Values Between Two Adjoining Holes in Section (Rule of Nearest Points).

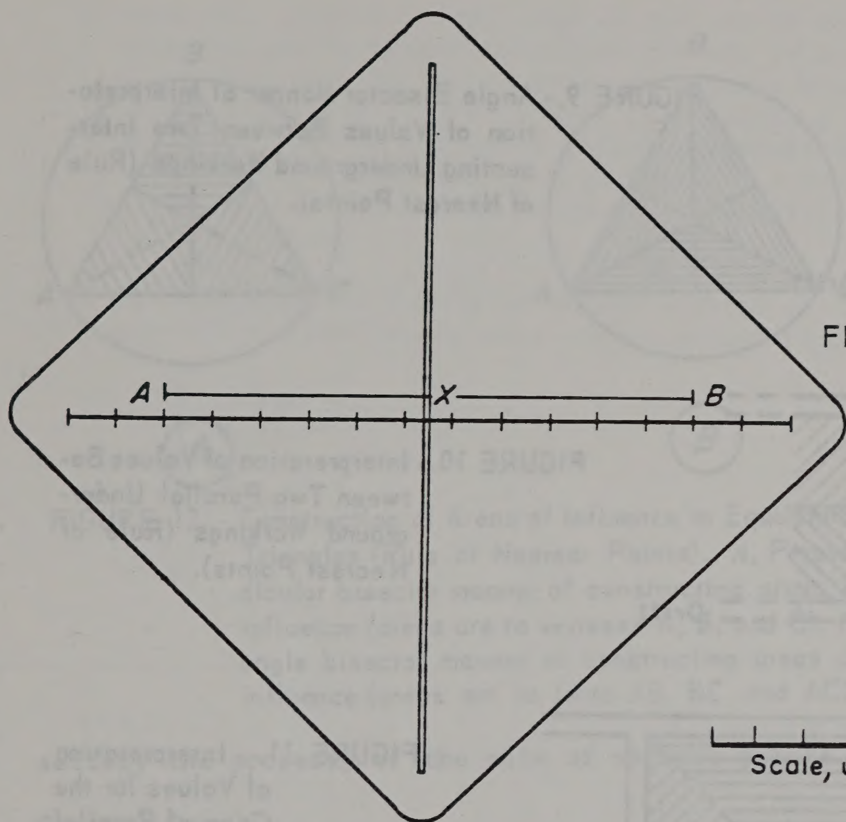


FIGURE 8. - Template or Guide to Find the Midpoint Between Two Points (Rule of Nearest Points).

application varies due to the type and distribution of workings, and whether these workings are presented on maps in the form of dots or lines. Several of the more common cases are described below.

Case of Two Underground Intersecting Workings

When a drift and a raise intersect in a plane of a map, the areas of equal influence of each working are found by bisecting the angle between them. Any point on the bisector is equidistant from both workings, and any point within each area of influence is nearer to the adjoining working than to the other (fig. 9).

Case of Two Parallel Workings

A line constructed equidistant between two parallel workings, such as trenches, drifts, crosscuts, raises, and drill holes, will divide the intervening area into two areas of equal influence, each satisfying the property of nearest points (fig. 10).

General Case of Underground Workings

When a block is developed by drifts and raises on all four sides, the area between them is divided into four areas of influence by a combination of angle bisectors and parallel lines (fig. 11). To satisfy the property of

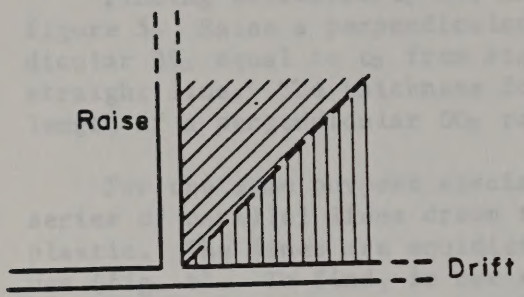


FIGURE 9.- Angle Bisector Manner of Interpretation of Values Between Two Intersecting Underground Workings (Rule of Nearest Points).

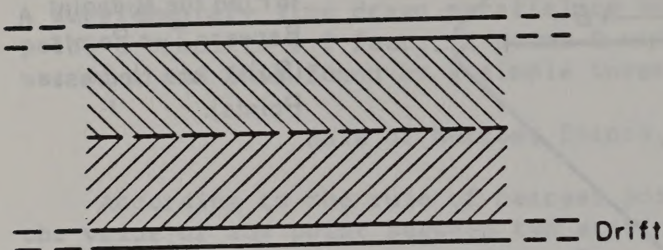


FIGURE 10.- Interpretation of Values Between Two Parallel Underground Workings (Rule of Nearest Points).

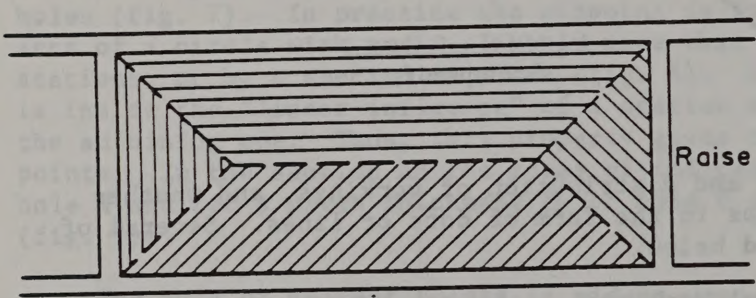


FIGURE 11. - Interpretation of Values for the Case of Parallel and Intersecting Workings (Rule of Nearest Points).

equal influence for each pair of workings in the block, only the illustrated construction is possible.

Case of Equilateral Triangle

In an equilateral triangle the areas of influence of each vertex are found by constructing perpendicular bisectors from the midpoint of each side (fig. 12A). The intersection of the bisectors is equidistant from the vertexes; it is the center of a circle passing through the three vertexes.

By constructing angle bisectors, different shaped areas are formed (fig. 12B) in comparison with perpendicular bisectors. This manner of dividing a triangle is called the rule of gravity.

Case of an Obtuse Triangle

In an obtuse triangle, the angle bisectors will divide the figure into three areas different in shape, but equal in size (fig. 13). The point of

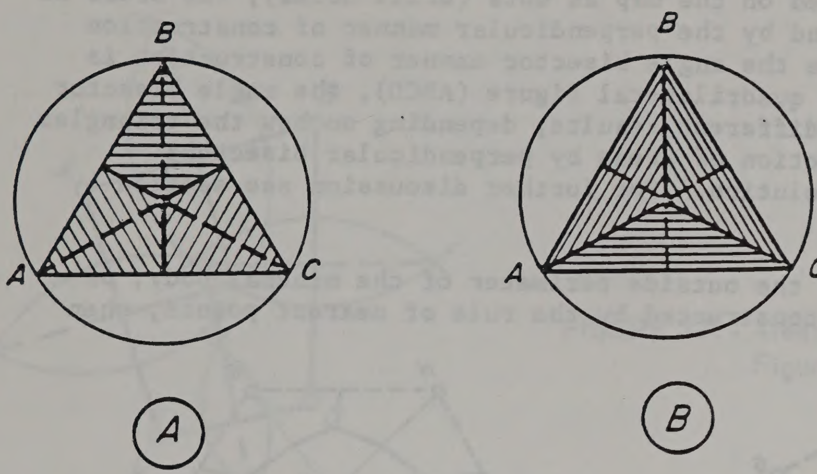


FIGURE 12. - Construction of Areas of Influence in Equilateral Triangles (Rule of Nearest Points). *A*, Perpendicular bisector manner of constructing areas of influence (areas are to vertexes A, B, and C); *B*, angle bisector manner of constructing areas of influence (areas are to lines AB, BC, and AC).

intersection of the angle bisectors, or the center of the gravity of the triangle, is much closer to the vertex of the obtuse angle than to the other vertexes. Therefore, the center as well as other points of the triangle, are inconsistent with the rule of nearest points.

Areas of influence constructed by perpendicular bisectors in an obtuse triangle are of different sizes, but the vertexes are equidistant from the intersection of the perpendiculars. The areas of influence of such a construction satisfy the property of the rule of nearest points.

General Case

Both manners of construction of areas of influence are used in computing reserves. The angle bisector manner is limited to workings presented on plans and sections as lines, such as intersecting underground workings (fig. 11).

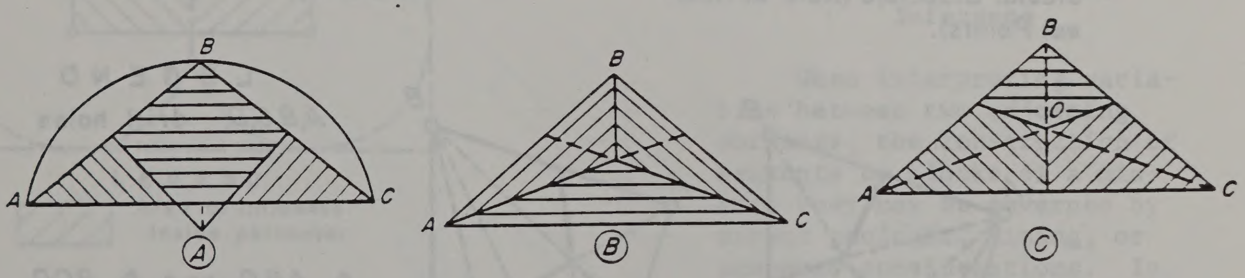
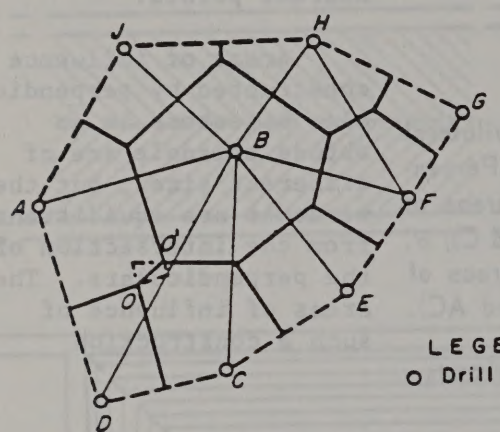


FIGURE 13. - Perpendicular Bisector Versus Angle Bisector Manner of Constructing Areas of Influence in Obtuse Triangles. *A*, Perpendicular bisector manner of constructing areas of influence for vertexes A, B, and C; *B*, angle bisector manner of constructing areas of influence for lines AB, BC, and AC; *C*, angle bisector manner of constructing areas of influence for vertexes A, B, and C. The last method is incorrect from the standpoint of rule of nearest points (O is closer to B than to A and C).

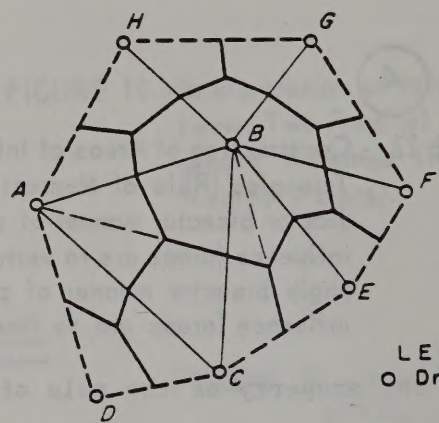
When the workings are presented on the map as dots (drill holes), the areas of influence of each one are found by the perpendicular manner of construction (fig. 14). In the latter case the angle bisector manner of construction is incorrect (fig. 15). For the quadrilateral figure (ABCD), the angle bisector construction may produce two different results, depending on how the triangles are drawn (fig. 16). Construction of areas by perpendicular bisectors (fig. 17) produces only one solution. For further discussion see section entitled "Method of Polygons".

An area of influence for the outside perimeter of the mineral body, or for an isolated hole, can be constructed by the rule of nearest points, when



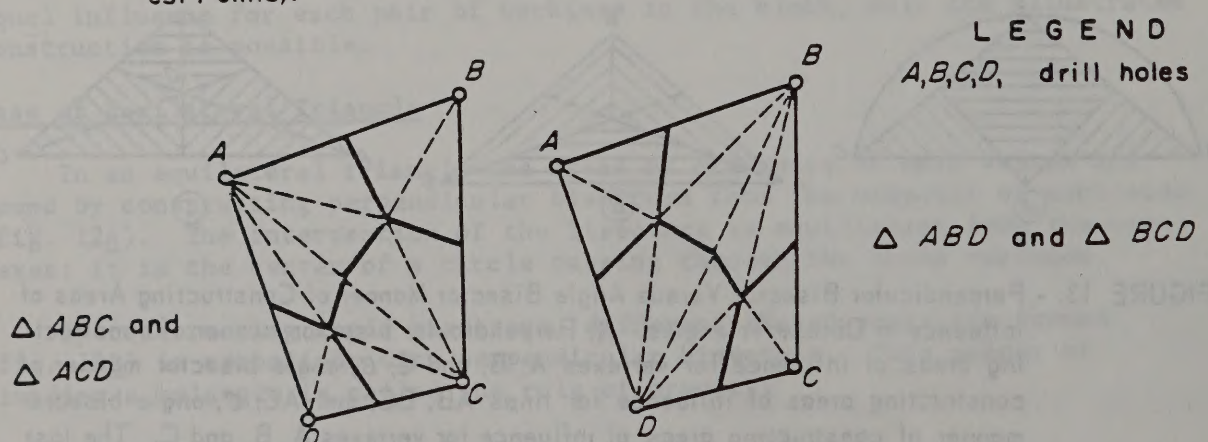
Case of exploration by vertical drill holes.

FIGURE 14. - Correct Construction of Areas of Influence (Polygons) by Perpendicular Bisectors (Rule of Nearest Points).



Case of exploration by vertical drill holes

FIGURE 15. - Incorrect Consfruction of Polygons by Angle Bisectors (Rule of Gravity).



Two solutions — depending on construction of triangles
(both incorrect)

FIGURE 16. - Areas of Influence for Quadrilateral Figures (Rule of Gravity).

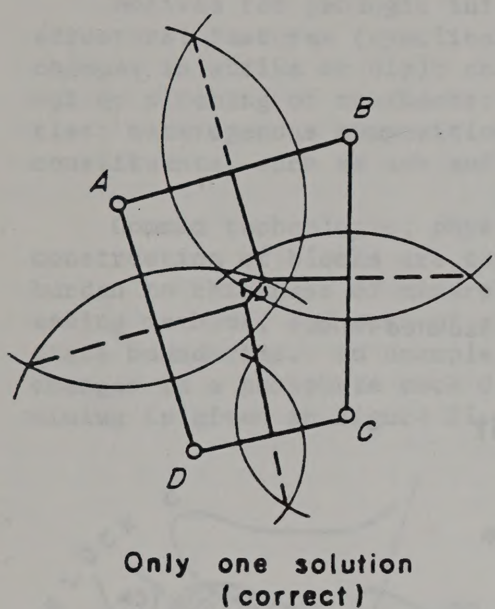


FIGURE 17. - Areas of Influence for Quadrilateral Figures (Rule of Nearest Points).

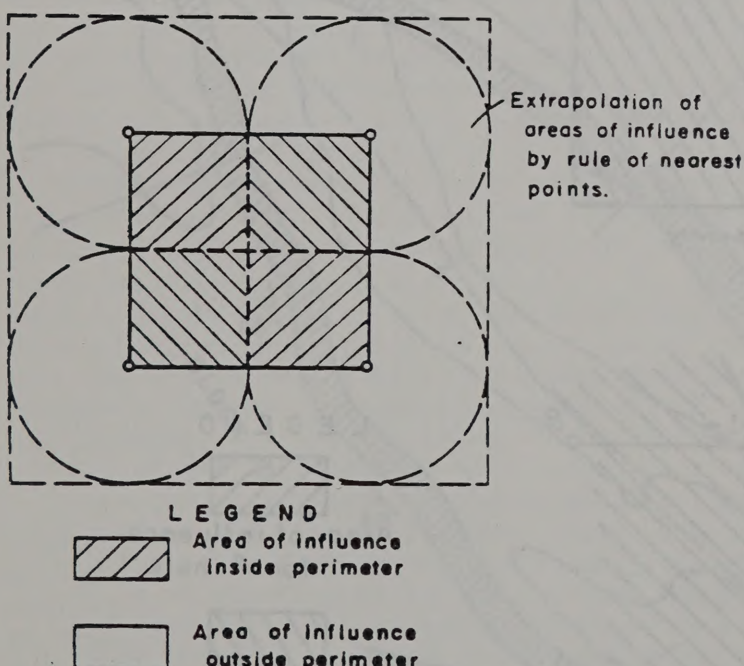


FIGURE 18. - Areas of Influence for a Square Block (Rule of Nearest Points).

a "standard" mean radius of influence for a certain category of reserves and type of deposit is accepted. Such areas may be constructed by a circle equal to the standard radius of influence (figs. 18 and 19).

Geologic and Mining Inference

When interpreting variables between two adjoining workings, the construction of segments or blocks of a mineral body may be governed by direct geologic, mining, or economic considerations. In a simple case of two drill holes with corresponding thicknesses t_1 and t_2 of ore and a prominent vertical fault between them, the sphere of influence (areas of influences for section and plan) may be assigned on basis of geologic interpretation, as illustrated in figure 20; that is, the thickness t_1 is consigned to ore between the hole A and the fault, and the thickness t_2 between the fault and hole B.

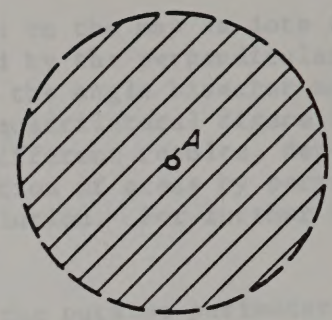


FIGURE 19. - Area of Influence for an Isolated Hole.

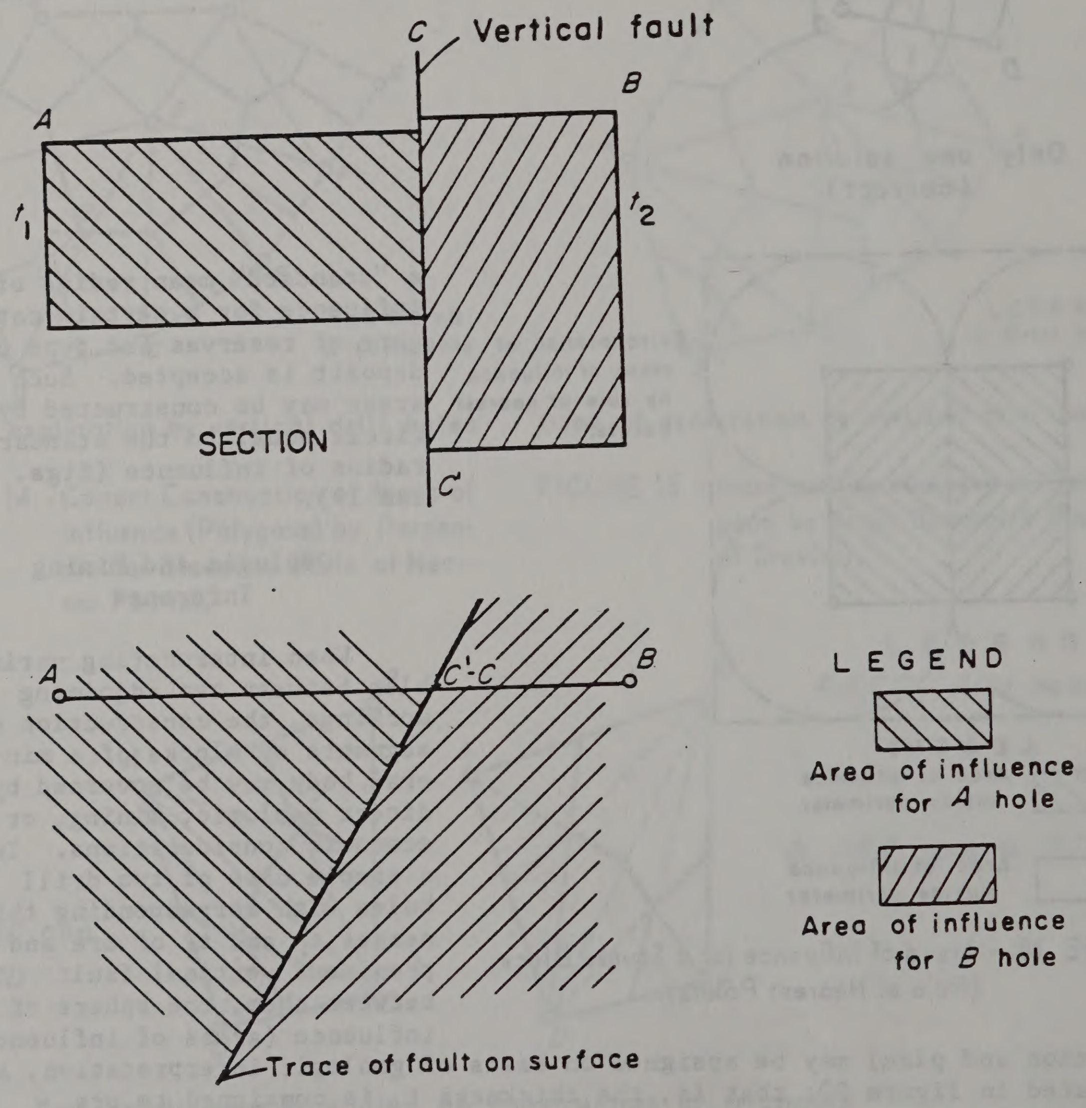


FIGURE 20. - Geologic Interpretation of Areas of Influence Between Two Adjoining Stations.

Motives for geologic inference include natural geologic boundaries due to structural features (synclines, anticlines, faults, or other dislocations, changes in strike or dip); changes in character of mineralization; thinning out or pitching of oreshoots; zoning; weathering; different physical properties; heterogenous composition; varied alteration; and presence of detrimental constituents, such as ash and sulfur in coal.

Common technologic, physiographic, and economic grounds for inference in construction of blocks are topography, thickness of overburden, ratio of overburden to thickness of mineral body, depth, water level, mining methods, processing methods, and cost of extraction; also property, section, township, and state boundaries. An example of block construction on the basis of structural changes in a phosphate rock deposit and availability of ore for open pit mining is given in figure 21.

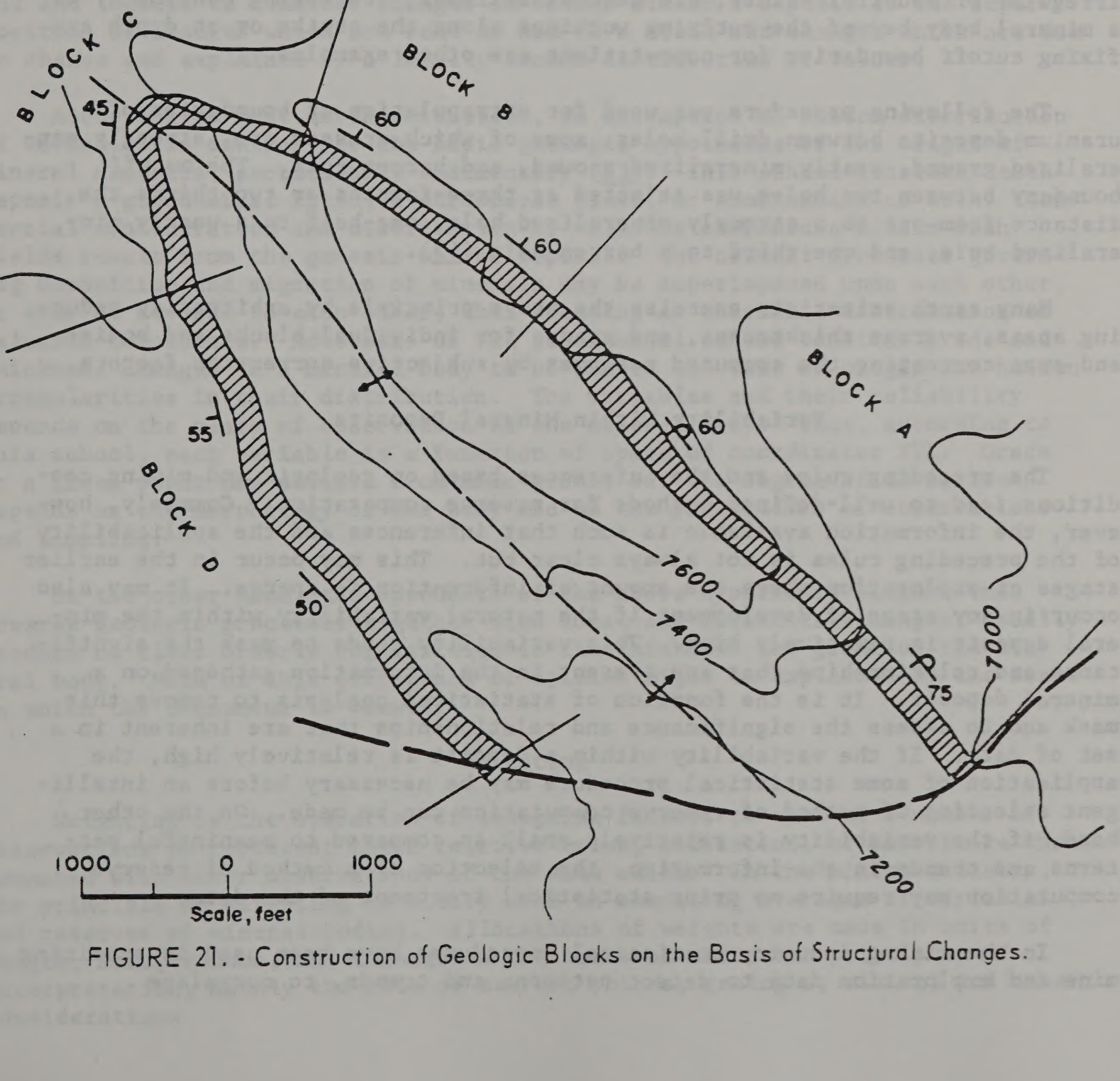


FIGURE 21. - Construction of Geologic Blocks on the Basis of Structural Changes.

Rule of Generalization

The rule of generalization is also known as the empirical method and, in its extreme, as the rule of thumb. It is used frequently for interpretation of exploration data. In contrast with the more objective interpretations described previously, such a rule is used rather arbitrarily. It is often adapted for lack of other criteria on the basis of limited experience, or as a matter of judgment and generally reflects past experience and opinion.

In many cases, the use of the rule is justifiable and unavoidable. Adapting a definite weight factor for reserve computations from other similar mineral deposits is probably the most common example. Selecting specific limits for the size of blocks in classifying reserves by categories for certain mineral deposits or assuming factor values for reserves on the basis of production data, rather than directly from widely spaced drill holes with irregular or doubtful values, are generalizations. Projecting continuity of a mineral body beyond the outlying workings along the strike or at depth and fixing cutoff boundaries for computations are other examples.

The following procedure was used for extrapolation of boundaries of uranium deposits between drill holes, some of which crossed ore, strongly mineralized ground, weakly mineralized ground, and barren rock. The cutoff boundary between two holes was selected as three-fourths or two-thirds the distance from ore to a strongly mineralized hole, one-half to a weakly mineralized hole, and one-third to a barren hole (63).

Many earth scientists exercise the above principle by arbitrarily reducing areas, average thicknesses, and grades for individual blocks and bodies and even correcting the computed reserves by subjective correction factors.

Variability Within Mineral Deposits

The preceding rules and the inferences based on geologic and mining conditions lead to well-defined methods for reserve computation. Commonly, however, the information available is such that inferences and the applicability of the preceding rules is not always clear cut. This may occur in the earlier stages of exploration where the amount of information is sparse. It may also occur in any stage of development if the natural variability within the mineral deposit is relatively high. This variability tends to mask the significance and relationships that are present in the information gathered on a mineral deposit. It is the function of statistical analysis to remove this mask and to assess the significance and relationships that are inherent in a set of data. If the variability within a deposit is relatively high, the application of some statistical procedure may be necessary before an intelligent selection of method of reserve computation can be made. On the other hand, if the variability is relatively small as compared to meaningful patterns and trends in the information, the selection of a method of reserve computation may require no prior statistical treatment of the data.

In the mining industry statistical procedures have been used in examining mine and exploration data to detect patterns and trends, to correlate

variables, and "to develop numerical data from which the reliability of estimates can be assessed" (32). Partly because these procedures are well adapted to electronic computing, they have been useful in obtaining the maximum amount of information from sparse exploration data and in handling large amounts of data from operating properties. The application of statistical methods to the results of sampling and reserve computations has been discussed in many publications (4-5, 15-18, 22, 62).

Statistical procedures are generally useful in isolating changes or variability that is due to chance from changes that are "real." The reliability of estimates is developed from the variability that is due to chance. In data obtained from mineral deposits, this variability may arise from a random distribution of values within the region considered. As the region considered is enlarged, "real" changes or trends invariably appear and the values within this enlarged region can no longer be considered random. It is generally useful and logical to consider changes in values within a deposit as the superposition of changes which are real or due to a trend and changes which are due to chance and explained by a locally random distribution of values.

According to most earth scientists, an assumption of random distribution of variables is contrary to the basic geologic hypothesis of the origin of mineral deposits, particularly sedimentary (53). This school considers each deposit a geochemical field, a structural field, or combination of both. Commercial concentration and distribution of valuable components within such fields result from the genesis of the deposit. The natural processes governing deposition and migration of minerals may be superimposed upon each other, or even be adverse to each other, thus creating an intricate distribution of valuable components. Advocates of the geochemical school consider grade and thickness changes in a mineral body to be due to the mode of origin and hidden irregularities in their distribution. The variables and their reliability depends on the place of observation in the mineral body. Thus, according to this school, each variable is a function of space of coordinates XYZ. Grade at a given point may deviate from mean grade, but the degree of deviation depends on the morphology of the body and on the particulars of observations and sampling.

On the other hand, the opponents of the above hypothesis believe that adverse geologic processes together with local and accidental changes, usually produce no clear orderly regularities in the thickness and grade of the mineral body. Much of this difference of opinion might be explained by the scale on which the phenomena is viewed.

Weighting

Weighting is the operation of assigning factors to each of a number of observations to represent their relative value, allocation, or importance when compared with other observations of the same suite. In the mining industry, the principle of weighting is widely used in computing averages of variables and reserves of mineral bodies. Allocations of weights are made in units of length, area, volume, and tonnage on the basis of different principles of interpretation, mainly the rule of nearest points, geologic, mining, and other considerations.

The use of weighting in each particular case depends on the analysis of exploratory data. In sectional sampling, across a wide mineral body, weighting may be compulsory for computation of average grade over the entire width of a vein with different metal values near the hanging wall and footwall. In all methods of reserve computations, the principle of weighting is applied to individual blocks of different sizes to determine average thickness and average grade of the entire deposit.

In certain cases, weighting by an area of influence is not appropriate. For a region within which the values are randomly distributed, no sample by definition has an area of influence; hence, weighting samples within this region by an area of influence is not logical in obtaining an average for the region. Thus, the rule of nearest points is not applicable for this case. However, this does not mean the samples from such a region should not be weighted for some other reason when computing the region average.

Application

All the above principles of interpretation are used in valuation of mineral deposits. A study of the common methods of computations discloses that block construction is usually based on one definite principle, and other principles, often secondary in importance, applied as supplementary operations (table 1). The principles of statistical analysis, weighting, and generalization are used in all conventional methods.

TABLE 1. - Principles of interpretation of exploration data used in construction of blocks and reserve computations

(XXX - Predominant; XX - Supplementary but influential; X - Secondary)

Reserve computations: Conventional methods and modifications	Intrinsic		Analytical		Rule of generalization
	Geologic	Mining and economics	Rule of gradual changes	Rule of nearest points	
Average factors and area:					
Analogous..................	XXX	XX	-	-	X
Geologic blocks..........	XXX	X	-	-	XX
Mining blocks..............	XX	XXX	-	-	X
Cross sections:					
Standard..................	XX	XX	XXX	-	X
Linear....................	XX	XX	-	XXX	X
Isolines..................	XX	XX	XXX	-	X
Triangular prisms..........	X	-	XXX	-	XX
Polygonal prisms...........	X	-	-	XXX	XX

The leading principle in average factors and area methods is based on geologic criteria. Mining, economic, and to lesser extent, geologic criteria

support the mining blocks method. The rule of gradual changes is basic to the method of triangular prisms and the rule of nearest points to the method of polygonal prisms. The rule of gradual changes is the predominant principle in the standard and isolines cross-section methods, and the rule of nearest points is used in the linear cross-section method.

Computations

Basic Parameters

The basic parameters for computing reserves of a mineral deposit include thickness and area - quantitative indicators of form, size, and volume of the mineral body; grade - the qualitative indicator of values and their distribution in the deposit; and weight factor or specific gravity - indicator for tonnage computations.

In most deposits thickness and grade vary from place to place in greater degree than the weight factor. For simplicity the latter is considered constant in this report.

Thickness and Area

Measurements of the thickness of a mineral body are taken directly by a series of observations, scaled from maps and sections, or computed, and then arithmetically averaged,

$$t_{av} = \frac{t_1 + t_2 + t_3 + \ldots + t_n}{n}. \qquad (10)$$

Area is measured directly from maps by planimetering, by the use of specially constructed templates, by geometric computations, and indirectly by computing from survey data.

Planimetering

At least two planimeter readings taken in opposite directions are necessary to achieve correct results. If these readings vary by less than 2 percent, the average is accepted as true. The scale of the selected maps should meet the accuracy requirements of the smallest area measured.

Templates

Templates may be of square pattern, where each square has a certain unit-area value; of dotted pattern, where each dot is the center of a unit of equal area; or, of parallel lines pattern with a series of equidistant lines drawn to scale (figs. 22, 23, 24). The use of templates with the first two patterns is self-evident. In the case of the parallel lines template the lengths of all lines within the mineral body are totaled; the sum of lengths multiplied by the unit value of the scale equal the total area. Two different positions of a template are taken for precise measurements and the average accepted as

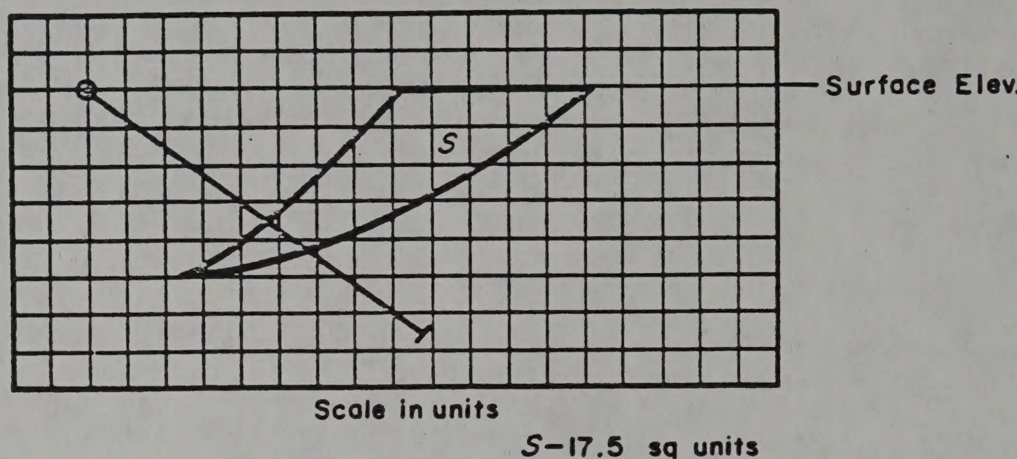


FIGURE 22. - Square Pattern Template.

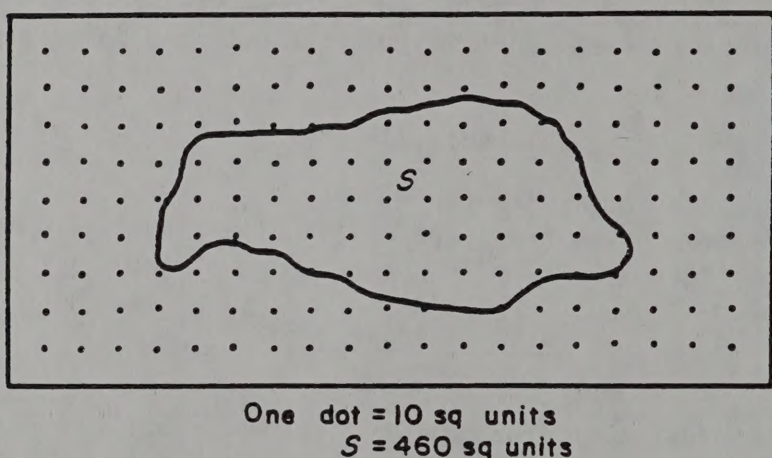


FIGURE 23. - Dotted Pattern Template.

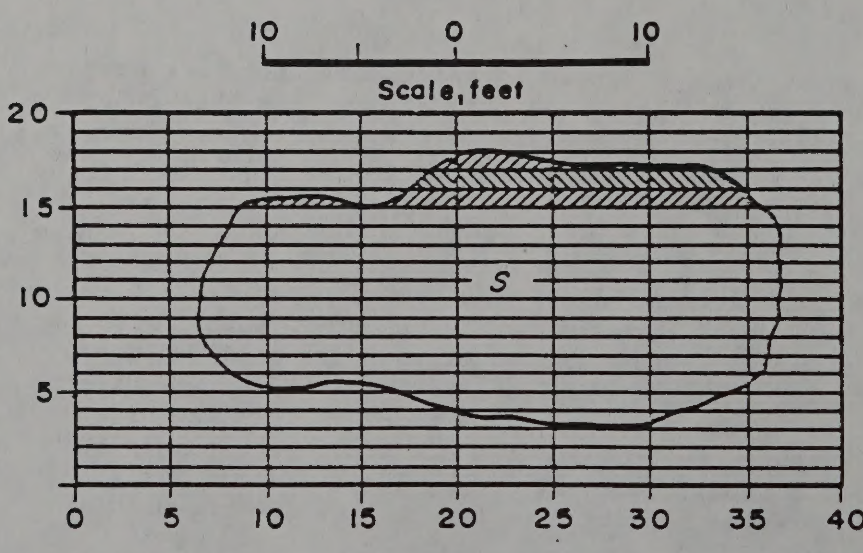


FIGURE 24. - Parallel Lines Template.

the true area. In practice, the square pattern is used when the area is 50 units or less.

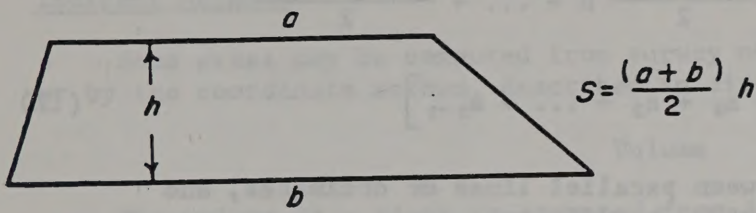
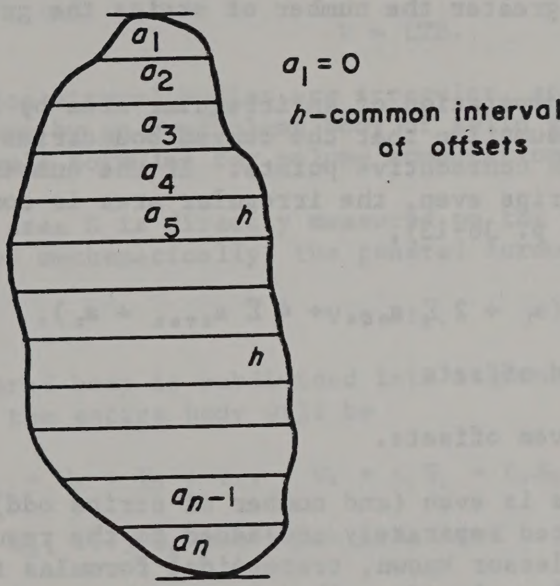


FIGURE 25. - Trapezoid Formula.



$$S = \frac{(a_1 + a_2)}{2} h + \frac{(a_2 + a_3)}{2} h + \ldots + \frac{(a_{n-1} + a_n)}{2} h$$

$$= h \left[\frac{(a_2 + a_n)}{2} + a_2 + a_3 + \ldots + a_{n-1} \right]$$

FIGURE 26. - Trapezoidal Rule.

Geometric Computations

Irregularly shaped areas may be divided into simple geometric figures; that is, triangles, squares, tetragons, and trapezoids. The dimensions of each figure can be scaled from maps or deduced from survey notes and the area computed. The total area is equal to the sum of the calculated figures. The most common formulas for plain figures, triangle, square, rectangle, and parallelogram are well known. Formulas for the trapezoid follow.

Trapezoid Formula. - An area of a simple trapezoid is

$$S = \frac{(a + b)}{2} h, \tag{11}$$

where a and b are parallel sides of the figure and h the perpendicular distance (fig. 25).

Trapezoidal Rule. - An irregular area may be subdivided into an even number of trapezoidal figures by a series of equidistant parallel lines, or ordinates (fig. 26). Assuming that the boundaries of the strips between the ordinates are straight lines, the entire irregular area may be computed by the trapezoidal rule,

$$S = \frac{(a_1 + a_2)}{2} h + \frac{(a_2 + a_3)}{2} h + \ldots + \frac{(a_{n-1} + a_n)}{2} h$$

or

$$S = h \left[\frac{(a_1 + a_n)}{2} + a_2 + a_3 + \ldots + a_{n-1} \right] \qquad (12)$$

where h is a common interval between parallel lines or ordinates, and $a_1, a_2, \ldots, a_n$ are the lengths of each ordinate.

It is obvious that the greater the number of strips the greater is the precision of the formula.

Simpson's Rule. - The computation of an irregular area by Simpson's rule (fig. 27) is based on the assumption that the curved boundaries of each strip are parabolas passing through consecutive points. If the number of offsets are odd and the number of strips even, the irregular area is computed by Simpson's formula (42, v. 2, p. 36-13),

$$S = \frac{1}{3} h \,(a_1 + 2\,\Sigma\, a_{odd} + 4\,\Sigma\, a_{even} + a_n), \qquad (13)$$

where $\Sigma\, a_{odd}$ - the sum of odd offsets

$\Sigma\, a_{even}$ - the sum of even offsets.

If the number of offsets is even (and number of strips odd), one of the end-area trapezoids is computed separately and added to the results computed by the formula. Other, and lesser known, trapezoidal formulas for determining area are Durand's and Weddle's rules described in engineering handbooks.

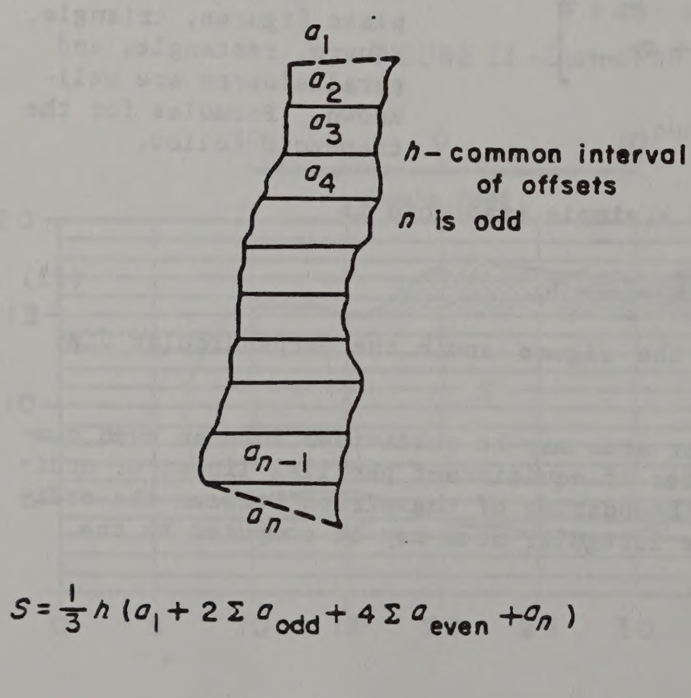


FIGURE 27. - Simpson's Rule for Determining Areas.

Indirect Methods

Some areas may be computed from survey notes by double meridian distances or by the coordinate method, described in civil engineering handbooks.

Volume

The volume of a block is computed from direct or indirect measurements of length (L), breadth (B), and thickness (T) by the parallelepiped formula,

$$V = LTB. \tag{14}$$

In practice mineral bodies are irregular, and it is necessary to substitute true volume by an equivolume body of solid geometric configuration for the use of simple formulas for volume computations.

When the area S is directly measured on the map and the average thickness t_{av} is computed methematically, the general formula for the deposit is

$$V = St_{av}. \tag{15}$$

If a mineral body is subdivided into segments or blocks for computations, the volume of the entire body will be

$$V = V_1 + V_2 + V_3 + \ldots + V_n = t_1 S_1 + t_2 S_2 + t_3 S_3 + \ldots + t_n S_n , \tag{16}$$

where S_1, S_2, S_3, ..., S_n are block areas and

t_1, t_2, t_3, ..., t_n are average thicknesses of individual blocks.

Various methods of block construction are discussed in part 2 of this report. It is obvious that the substitution error of the true volume of a mineral body with auxiliary blocks depends on the knowledge of the form and size of the body. In addition, the accuracy of computations depends on the number of blocks, variations in the size of blocks, and precision obtained by the formulas.

Weight

Tonnage Factors

Conversion of volume to tonnage of raw mineral material (ore, rock, and coal) varies, depending on the system of measures used. Common formulas used in computing tonnages are

$$Q = \frac{V}{F} \text{ and } Q = Vf. \tag{17}$$

In the first formula F is the volume-tonnage factor and is usually expressed in cubic feet per ton. In the second formula f is the tonnage-volume factor and is usually expressed in weight-units per cubic foot.

Both weight factors are interrelated and are determined on the basis of past production, experimental mining, or adapted from similar deposits. They also may be determined by measuring excavations, or by special laboratory tests. Techniques of these determinations are described in several publications (36, 40, 42). In some cases tonnage factors may be computed from the mineral composition after corrections for porosity and moisture content of raw material are made.

In many deposits the weight factors vary substantially owing to the mineral and grade composition. The relationship between weight and grade often may be expressed graphically; thus, weight factors can be determined for appropriate grade of each individual block.

The ore tonnage of the entire body is determined by formula,

$$Q = Q_1 + Q_2 + Q_3 + \ldots + Q_n = V_1 f_1 + V_2 f_2 + V_3 f_3 + \ldots + V_n f_n . \quad (18)$$

Specific Gravity

Conversion of volume to tonnage (metric) is made by the formula,

$$Q = VD, \quad (19)$$

where D is specific gravity or density of raw mineral material.

The specific gravity of the mineral matter can be determined by direct tests of dried and crushed samples. Specific gravity of rock in place, or "rock specific gravity-natural" may be expressed by

$$D_{nat} = \frac{D_s \, (1 - P_o)}{(1 - M_o)}, \quad (20)$$

where D_s is the specific gravity of the mineral matter determined by tests of crushed and dried rock.

P_o is porosity in percent pore space to unit of volume.

M_o is moisture in percent weight loss upon drying. Mead offers a convenient diagram for the English system of weights and measures, showing the influence of porosity, moisture, and specific gravity on the tonnage factor (36).

Specific gravity may be calculated theoretically as an average of the specific gravity of all of the minerals in the deposit, or according to the weighted average percent of each mineral in the rock. For convenience, the calculated deviations of specific gravity for various grades may be presented graphically.

Conversion Formulas

The volume-tonnage factor, F, is computed from specific gravity by the following formulas (42, v. 2, p. 25-20):

$$\text{For short ton } F_{s.t.} = \frac{2{,}000}{62.5\ D}\ ft^3/s.t., \text{ and}$$

$$\text{For long ton } F_{L.t.} = \frac{2{,}240}{62.5\ D}\ ft^3/L.t., \tag{21}$$

where 62.5 lb is the weight of 1 cubic foot of water at 4° C.

The tonnage-volume factor, f, is computed by

$$f = \frac{2{,}000}{F_{s.t.}} \text{ or } \frac{2{,}240}{F_{L.t.}} \text{ or } 62.5\ D. \tag{22}$$

Volume in cubic feet can be converted to cubic meters by multiplying by 0.028 or dividing by 35.3. Metric tons are converted to short tons by multiplying 1.102 or to long tons by 0.984 (Appendix A).

When the density of a particular body varies appreciably from one place to another owing to the relative amounts of minerals with wide ranges in specific gravities, such as galena and iron oxide versus quartz, more accurate results are obtained by

$$Q = V_1 D_1 + V_2 D_2 + V_3 D_3 + \ldots + V_n D_n \tag{23}$$

where $D_1, D_2, \ldots, D_n$ are specific gravities of separate blocks $V_1, V_2, \ldots, V_n$.

Grade

Grade computation of a mineral body is a critical and important operation that can be done by various formulas:

1. Simple arithmetic mean (unweighted).
2. Weighted per width or thickness.
3. Weighted per width and length, or area.
4. Weighted by frequency of occurrence.
5. Weighted by the square of the frequency.
6. Weighted by frequency and assay (35).

The problems of sampling of various types of mineral deposits and methods of statistical analysis used in evaluating exploration data, and in computing average grade of workings, blocks, and commercial portions of bodies, are beyond the scope of this paper.

Generally, average grade of a mineral body is computed using conventional methods of reserve computations; the formulas used are

Type of problem	Assumption	Equation	Equation No.
Arithmetic average.	All blocks are equal in area, thickness, and weight factor.	$c_{av} = \frac{c_1 + c_2 + c_3 + \ldots + c_n}{n}.$	(24)
Thickness--weighted average.	All blocks are equal in area and have the same weight factor.	$c_{av} = \frac{c_1 t_1 + c_2 t_2 + c_3 t_3 + \ldots + c_n t_n}{t_1 + t_2 + t_3 + \ldots + t_n}.$	(25)
Area--weighted average.	All blocks have constant thickness and weight factor, but different areas.	$c_{av} = \frac{c_1 S_1 + c_2 S_2 + c_3 S_3 + \ldots + c_n S_n}{S_1 + S_2 + S_3 + \ldots + S_n}.$	(26)
Volumetric average (volume-weighted average).	Weight factors of all blocks are the same.	$c_{av} = \frac{c_1 V_1 + c_2 V_2 + c_3 V_3 + \ldots + c_n V_n}{V_1 + V_2 + V_3 + \ldots + V_n}.$	(27)
Gravimetric average (tonnage-weighted average).	Tonnages and grades of blocks are different.	$c_{av} = \frac{c_1 Q_1 + c_2 Q_2 + c_3 Q_3 + \ldots + c_n Q_n}{Q_1 + Q_2 + Q_3 + \ldots + Q_n}.$ *	(28)

*Q_1, Q_2, Q_3, ..., Q_n are reserves of raw mineral material in individual blocks, in tons.

Reserves of valuable components are determined by formula

$$P = Q\, c_{av}, \tag{29a}$$

where P is the sum of reserves of each valuable component of individual blocks P_1, P_2, P_3, ..., P_n and Q is the sum of reserves of raw material Q_1, Q_2, Q_3, ..., Q_n. The average grade of deposit is determined by formula

$$c_{av} = \frac{P}{Q}. \tag{29b}$$

Errors

Accuracy Versus Precision

The terms "accuracy" and "precision" as related to reserve computations are defined in this report as follows: The variance between a single observation or a computed average of an element of a mineral body and its true value indicates the degree of accuracy or exactness of such observations or estimates. An accurate measurement is free from all errors. Precision indicates only the degree of fluctuation in a certain suite of variables with respect to their proximity to each other.

The distinction between accuracy and precision is well illustrated graphically in the following example for a suite of chemical analyses (1, 57).

1. Accurate and precise (fig. 28A).

2. Inaccurate but precise (figs. 28B and 28C).

3. Accurate but not precise (fig. 28D).

4. Inaccurate and not precise (figs. 28E and 28F).

The errors in reserve computations may be divided into three groups: errors of interpretation (often labeled geologic), technical, and analytical.

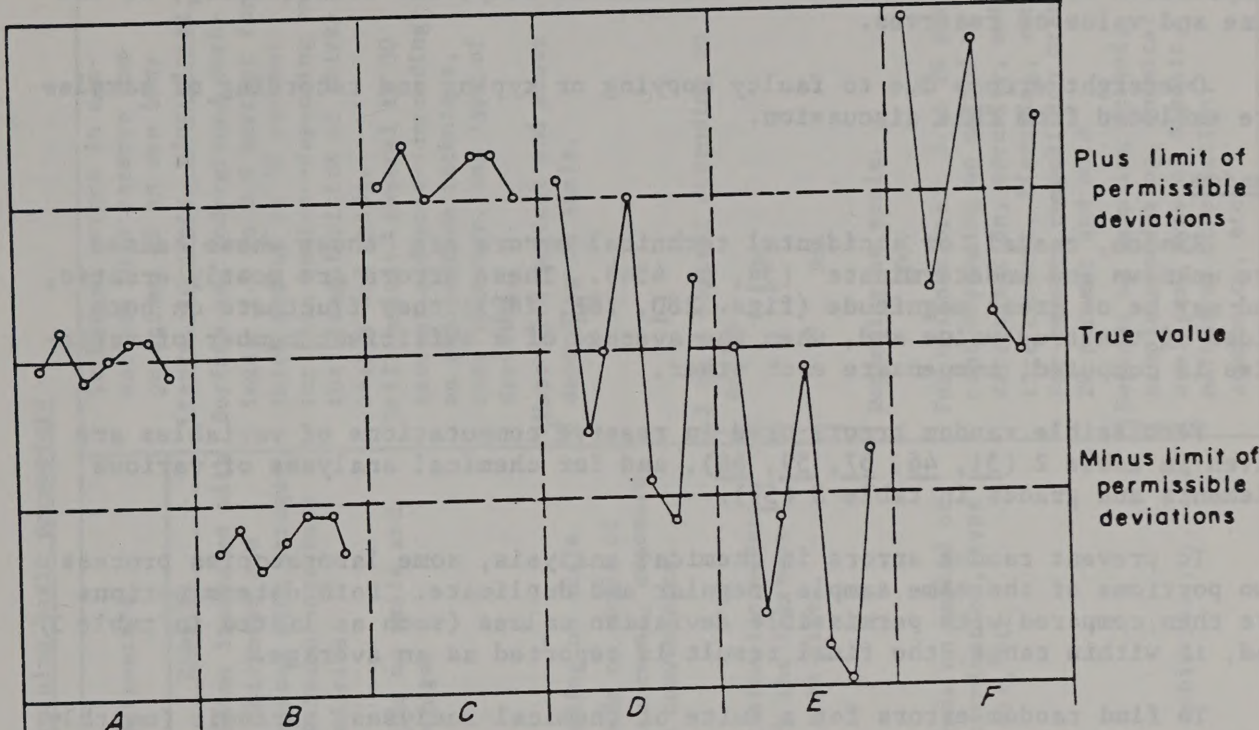


FIGURE 28. - Accuracy and Precision of Chemical Analyses. *A*, Accurate and precise; *B* and *C*, inaccurate, precise, large bias errors; *D*, accurate, not precise, large random errors; *E* and *F*, inaccurate, not precise, large random and biased errors.

Errors of Interpretation or Analogy

Errors of interpretation often called errors of analogy, representation, details, and geology are due to the accepted hypothesis of the origin of the deposit, assumption of geologic similarity to other deposits, interpretation or assumtpion of the uniform changes of the basic elements, and the continuity of the body along the strike and at depth. They are errors of judgment and, consequently, depend on the training and experience of the person appraising or conducting the investigation.

The results of exploration are generally disclosed by a series of plans and sections representing the mineral body in graphic form. Thus, the exactness of our knowledge of a definite mineral deposit depends on the correctness of the maps, which in turn, depends on the type of mineral deposit, kind and density of workings, and precision of all measurements and qualitative assays and tests.

Technical

Technical errors are those due to imperfections in instruments and techniques used in determining all variables. Errors, random, biased, or both, should be corrected to prevent downgrading or upgrading of individual observations, since erroneous variables influence interpretation of boundaries and

computation of basic parameters of the mineral body, and consequently, of the size and value of reserves.

Oversight errors due to faulty copying or typing and recording of samples are excluded from this discussion.

Random

Random, casual, or accidental technical errors are "those whose causes are unknown and undeterminate" (54, p. 454). These errors are mostly erratic, and may be of great magnitude (figs. 28D, 28E, 28F); they fluctuate on both sides of the true value and, when the average of a sufficient number of variables is computed, compensate each other.

Permissible random errors used in reserve computations of variables are given in table 2 (31, 46, 57, 59, 66), and for chemical analyses of various elements and grades in table 3 (57).

To prevent random errors in chemical analysis, some laboratories process two portions of the same sample, regular and duplicate. Both determinations are then compared with permissible deviation values (such as listed in table 3) and, if within range, the final result is reported as an average.

To find random errors for a suite of chemical analyses, periodic (monthly or quarterly) control analyses are made. These repeated samples are given in code in the amount of 3 to 5 percent of the total number of samples but not less than 30 (67). The magnitude of random error between control and regular analyses should not exceed the permissible limits adopted by the laboratory.

Biased

Biased or systematic errors are "those which affect all measurements alike" (54, p. 454). They are due to imperfections of instrument, equipment, and accepted techniques of observations. For chemical analyses such errors may be due to inexperienced personnel, inferior quality reagents, and selection of an improper method for a given sample. Most likely the errors are in one direction; that is, are persistently either more or less than the true value. Such errors affect the mean values because they are not compensating.

The presence and the magnitude of biased errors may be disclosed by special studies. To determine biased errors in chemical analyses, for example (67), control analyses of the same samples are made in a reputable laboratory by similar methods and procedures. Such outside laboratory control analyses are made for large projects at least twice a year; the number of analyses should be 3 to 5 percent of the total with a minimum of not less than 30 samples. If a substantial error is found and proved further by a third party, a correction factor should then be applied to all samples analyzed in the first laboratory.

TABLE 2. - Technical errors in determining basic parameters

	Random errors in determining individual parameters			Technical errors in connection with reserve computations of an ore body (range, accuracy, plus-minus)
	Original observations	Precision, plus-minus	Remarks	
Thickness...	Tape (for 1 meter)..	1.0 percent.........	Precision increases with ore thickness and decreases with irregularities of ore body boundaries.	Surface and underground workings: 2.0 to 3.0 percent for thick and up to 10 percent for thin bodies--depending on the irregularities of thicknesses and grade.
	From plans.........	Above 1.5 percent...	Depends on scale and drawings.	Drill holes: several to 30 percent and more--depending on exploration technique, core recovery, and type of deposits.
Length......	Surveying..........	0.5 percent and less	-	Maps--1.0 percent and above depending on scale.
	Drawing............	Up to 2.0 percent...	Depending on scale	
Angle.......	Brunton............	0.5 to 2.0 degrees..	Depends on angle of measurement, exposure, and convenience.	
Area........	Surveying data......	0.5 percent and less	Precision increases with the increase of area size.	2 to 3 percent depending on scale.
	Planimeter (100-400 cm^2).	1.5 to 0.3 percent (area).		Do.
	Template...........	Up to 3.0 percent...		Do.
	Maps:			
	Scale 1: 200	0.5 percent.........		Depending on scale.
	Scale 1:5000	0.25 percent........		Do.
Specific gravity or weight factor.	-	3 to 10 percent.....	Depends on method of determination, type of ore, etc.	For uniform ore: 3 to 20 percent depending on method of determination, porosity, moisture content, fissuring, etc. For complex composition ores: 20 percent and more.
Grade.......	Sample taking.......	Varied..............	-	Depends on sampling method, sample processing, chemical analyses, type of deposit, method of computation, commodity, etc.
	Sample processing...	1.0 percent.........	-	
	Chemical analysis...	1 to 20 percent and more.	See table 3...........	

TABLE 3. - Permissible average for random technical errors in chemical analyses

(The All-Union Committee on Mineral Reserves, U.S.S.R.)

Component and grade range of raw material, percent, except as noted	Permissible average error in percent to the grade determined (plus-minus)
Aluminum oxide (Al_2O_3):	
Above 20..........	2 - 4
5 - 20............	4 - 8
1 - 5.............	8 - 20
Antimony:	
Above 2...........	3 - 12
0.2 - 2...........	12 - 20
Arsenic:	
Above 2...........	1 - 5
0.5 - 2...........	5 - 7
Below 0.5.........	10
Barium sulfate ($BaSO_4$):	
Above 5...........	1 - 7
1 - 5.............	7 - 15
Beryllium:	
5 - 10............	3 - 5
0.1 - 5...........	5 - 10
0.01 - 0.1........	10 - 30
Bismuth:	
Above 0.6.........	5 - 15
0.2 - 0.6.........	15 - 20
Cadmium:	
Above 1...........	3 - 5
0.1 - 1...........	5 - 10
0.01 - 0.1........	10 - 30
Below 0.01........	30
Calcium oxide (CaO):	
Above 25..........	3 - 5
5 - 25............	5 - 10
1 - 5.............	10 - 25
Chromium:	
Above 10..........	1 - 3
1 - 10............	3 - 7
Below 1...........	7
Cobalt:	
Above 0.5.........	2 - 6
Below 0.5.........	6
Columbium:	
Above 10..........	3 - 5
1 - 10............	5 - 10
0.1 - 1...........	10 - 20
Below 0.1.........	20
Copper:	
Above 3...........	3 - 7
0.5 - 3...........	7 - 10
Below 0.5.........	10 - 15
Gold, g/m ton (1 g = 15.432 grains):	
Below 0.1 mm:[1]	
Above 64.......	2.5
16 - 64........	2.5 - 5
4 - 16.........	5 - 10
Below 4........	15
Below 0.6 mm:[2]	
Above 64.......	5
16 - 64........	5 - 10
4 - 16.........	10 - 20
Below 4........	25
Above 0.6 mm:[3]	
Above 64.......	7
16 - 64........	7 - 15
4 - 16.........	15 - 25
Below 4........	35
Iron:	
Above 30..........	1 - 2
10 - 30...........	2 - 4
5 - 10............	4 - 8
Iron oxide (FeO):	
Above 5...........	2 - 4
1 - 5.............	4 - 7
Lead:	
Above 15..........	2 - 4
6 - 15............	4 - 6
0.5 - 6...........	6 - 12
Below 0.5.........	12
Magnesium oxide (MgO):	
Above 5...........	3 - 10
1 - 5.............	10 - 20
Manganese:	
Above 5...........	2 - 4
1 - 5.............	4 - 7
0.05 - 1..........	7 - 20
Mercury:	
Above 2...........	4 - 7
0.25 - 2..........	7 - 15
Below 0.25........	15 - 20

See footnotes at end of table.

TABLE 3. - Permissible average for random technical errors in chemical analyses--Continued

(The All-Union Committee on Mineral Reserves, U.S.S.R.)

Component and grade range of raw material, percent, except as noted	Permissible average error in percent to the grade determined (plus-minus)	Component and grade range of raw material, percent, except as noted	Permissible average error in percent to the grade determined (plus-minus)
Molybdenum:		Tin:	
Above 1..........	2 - 5	Above 1..........	3 - 7
0.25 - 1.........	5 - 10	0.25 - 1.........	7 - 15
Below 0.25.......	10 - 20	0.05 - 0.25......	15 - 30
Nickel:		Titanium dioxide (TiO_2):	
1 - 5............	3 - 7	2 - 15...........	2 - 5
0.2 - 1..........	7 - 15	0.1 - 2..........	5 - 20
Below 0.2........	15	Tungsten trioxide (WO_3):	
Phosphorus:		Above 1..........	3 - 8
Above 3..........	3 - 7	0.25 - 1.........	8 - 15
0.03 - 0.3.......	7 - 15	0.05 - 0.25......	15 - 30
Silicon dioxide (SiO_2):		Vanadium:	
30 - 50..........	2 - 3	Above 0.5........	3 - 10
10 - 30..........	3 - 8	0.06 - 0.5.......	10 - 30
3 - 10...........	8 - 15	Zinc:	
Silver, grains per ton:		Above 25.........	2 - 3
Above 100........	1 - 3	10 - 25..........	3 - 6
30 - 100.........	3 - 5	0.5 - 10.........	6 - 15
10 - 30..........	5 - 12	Below 0.5........	15
Sulfur:		Zirconium:	
Above 20.........	1 - 2	Above 3..........	2 - 5
1 - 20...........	2 - 5	1 - 3............	5 - 10
0.05 - 1.........	5 - 10	0.1 - 1..........	10 - 15
Tantalum:		Below 0.1........	15 - 25
Above 10.........	3 - 5		
1 - 10...........	5 - 10		
0.1 - 1..........	10 - 20		
Below 0.1........	20		

[1] Samples with finely dispersed gold; mainly in sulfides.
[2] Samples with average grain size gold; in sulfides and quartz.
[3] Samples with large grain size, often visible, gold; mainly in quartz.

Source: Reference (57), table 7, pp. 67-68.

The correction factor may be computed as a ratio of the average grade of the control to the average grade of the regular analyses; that is

$$E = \frac{C_y}{C_x}. \tag{30}$$

The factor E is applied to regular samples to receive the correct results. For example, a regular suite of copper samples averaged 0.80 percent. Control samples averaged 1.0 percent copper. The correction factor is

$$E = \frac{1.00}{0.80} = 1.25.$$

The control analyses are 25 percent higher than the regular.

Analytical

Some analytical errors of reserve computations will be discussed in part 2. In general, the accuracy of computations increases with the number of blocks dividing the mineral body, provided the same accuracy is maintained in construction of each block. The error of a separate block may be high, but for a group of blocks representing the entire body the relative errors are balanced according to the law of averages;

blocks are equal in tonnage,

$$M_{av} = \frac{M_1 + M_2 + M_3 + \ldots + M_n}{N}; \text{ and} \qquad (31)$$

blocks are unequal in tonnage of valuable constituent

$$M_{av} = \frac{M_1 P_1 + M_2 P_2 + \ldots + M_n P_n}{NP}, \qquad (32)$$

where M_{av} is average relative error of mineral body, and M_1, M_2, M_3, ..., M_n are relative errors of individual blocks (in percent).

PART 2. - CONVENTIONAL METHODS

General Features and Classification

For reserve computations the mineral deposit, reduced and distorted by mapping, is converted to an analogous geometric body composed of one, several, or an aggregate of close-order solids, that best express the size, shape, and distribution of the variables. Construction of these blocks depends on the method selected. Some methods offer two or more manners of block construction, thus introducing subjectivity. In such a case a certain manner of construction is accepted as appropriate, preferably on the basis of geology, mining, and economics.

Numerous methods of reserve computations are described in the literature; some are only slight modifications of the most common ones. Depending on the criteria used in substituting the explored bodies by auxiliary blocks and on the manner of computing averages for variables, the conventional methods may be classified into four groups.

Group 1, average factors and area methods, embraces analogous and geologic blocks methods. Areas are delineated by geologic and, in part, by mining and economic criteria, and the basic elements (thickness, grade, and weight factors) are determined directly, computed, or taken from other portions of the same or similar deposits.

Group 2, mining blocks method, involves delineation of block areas by underground workings and by geologic and economic considerations; the factors for each block are computed in various ways. As the name implies the method is used mainly for extraction.

Group 3, cross-section methods, includes standard, linear, and isolines. The mineral body is delineated and the blocks are constructed on the basis of certain principles of interpretation of exploration data; the parameters of blocks and the entire body are determined in various ways.

Group 4, analytical methods, divides the mineral body graphically into blocks of simple geometric forms--triangular or polygonal prisms. The factors for each block are determined directly, computed as an arithmetic average, or in other ways.

Special studies of the usage of various methods were made in the U.S.S.R. Thousands of mineral deposits were explored and reserves computed and approved by the All-Union Committee on Mineral Reserves. The results for metal, nonmetal, and coal and oil-shale deposits for the years 1941-61 and for solid mineral deposits for years 1941-47, 1951, and 1954 are given in appendix B (table B-1) (57). The predominant methods were--

	Methods, percent		
	Average factors and area	Cross sections	Polygons
Coal and oil shale deposits................	69	-	30
Nonmetallic deposits.....	46	37	14
Ore deposits.............	37	48	14*

*Including mining blocks method.

It is also reported in the U.S.S.R. that in computing reserves the use of average factors and area and cross section methods together had increased from 30 (1941-47) to a total of 82 percent (1954) of all projects recorded (appendix B, table B-2).

A comparison of the use of various methods by 44 metal mines, described by Jackson and Knaebel in "Sampling and Estimation of Ore Deposits" (28) published in 1932, shows that the mining blocks and cross-section methods were predominant in the mining industry (table 4).

TABLE 4. - Usage of various methods for reserve computations for metal mines in U.S. (1932)[1]

Methods:	Percent	Methods:	Percent
Average factors and area...	20	Polygons..................	4
Mining blocks..............	42	Triangles.................	2
Cross sections.............	32	Total.................	100

[1] For 44 active mines described by Charles F. Jackson and John B. Knaebel. Sampling and Estimation of Ore Deposits, BuMines Bull. 356, 1932, pp. 125-249.

Average Factors and Area Methods

Assumptions and Characteristics Features

Average factors and area methods of reserve computations have been variously described as arithmetic average, weighted average, average depth and area, statistical, analogous (by analogy), geologic blocks, and general outline (27-28, 46, 57, 63). In this report these methods are discussed under the titles of analogous and geologic blocks.

Average factors and area methods are all based on the assumption that certain segments or blocks of the mineral body being considered are similar in geology and technology to sections previously studied, or to blocks or even bodies that have been explored or mined out. For computations the body is divided into segments or blocks constructed on the basis of geology, mining, and economic; that is, structure, thickness, grade, value, depth, and overburden. In some cases, the qualitative characteristics found in one part of the body may be accepted, for the purpose of computations, as representative of the block or the entire mineral body.

If the blocks are of equal size each observation and sample analysis has an equal influence in determining average factors. If the number of variables in a block are in sufficient quantity, average factors may be computed and studied by statistical analysis; on the other hand, the method of analogy may be used where only one observation is available. A number of segments or blocks with different controlling factors requires the use of the method of geologic blocks.

Formulas for computations of reserves range from simple to complex equations. Aside from the usual variables of grade, thickness, and density, more complex factors such as, tons recovered per unit of area, volume, or weight may be used.

Method of Analogy

Analogy is the inference that certain admitted partial resemblances probably imply further similarity. The method emphasizes qualitative similarity of the geology of a given block to an analogous and better known block of the same or similar body.

Variables for computations may be taken from a single or a number of observations, or computed from data gathered from the same or similar deposits. Such variables may be accepted as constant factors for other parts of the same body, other deposits, or even districts. When the geology of a given area or deposit is considered analogous to another area or deposit, a single observation may be adequate for reserve computations of a certain commodity. Reserves computed may belong to any category. The method is widely used in extraction operations when others are difficult to apply. In reserve computations of mica in pegmatites, for example, production records may be considered sufficient and accurate for assigning mica grade to the unmined portion of the vein below and between mined-out blocks.

Average Factors

The arithmetic average is the simplest variation of the analogous method. No auxiliary figures are constructed; thickness and grade are determined by a simple average of available data (fig. 29 and table 5). Grade may be determined also by thickness-weighting of individual grade observations from ore intersecting workings in the mineral body and even from adjoining parallel bodies and by extending the results to the unexplored block or to the entire mineral body (table 6). In case of numerous samples (observations) the average grade may be determined by statistical analysis (15-17). Reserve computations and the determination of the block-weighted average factors for the entire body is illustrated by table 7.

The formulas used are average thickness (formula 10), average grade (formula 24), thickness-weighted average grade (formula 25), volume of mineral body (formula 15), tonnage of raw material (formula 18), and tonnage of valuable component (formula 29a).

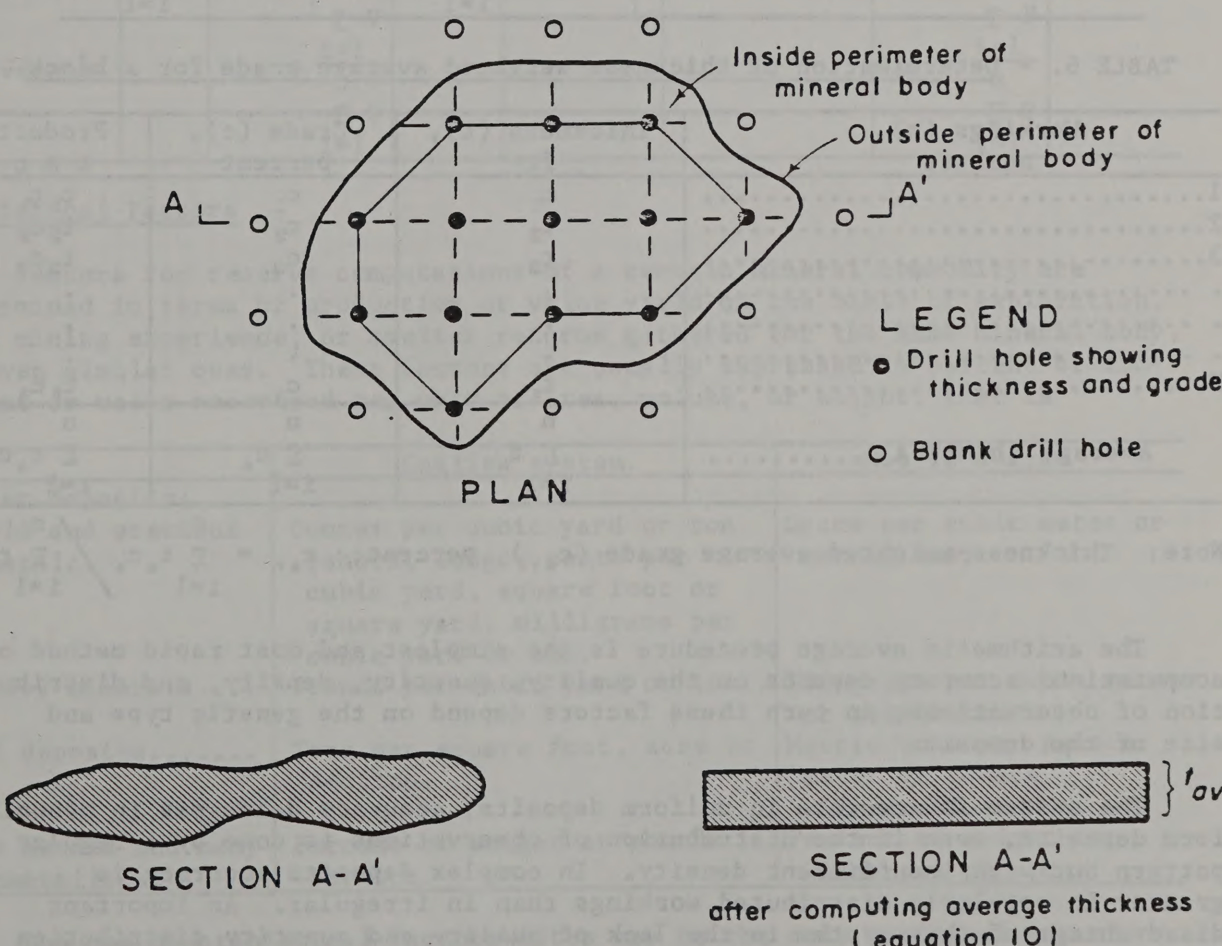


FIGURE 29. - Arithmetic Average Method of Computing Thickness. (For recapitulation of reserves for mineral body and determination of average grade, see table 5.)

TABLE 5. - Determination of average thickness and average grade for a block by arithmetic average procedure

Workings (n), numbers	Thickness (t), ft	Grade (c), percent
1..................................	t_1	c_1
2..................................	t_2	c_2
3..................................	t_3	c_3
. ...	.	.
. ...	.	.
. ...	.	.
N..................................	t_n	c_n
Total........................	$\sum_{i=1}^{n} t$	$\sum_{i=1}^{n} c$
Average........................	$t_{av} = \sum_{i=1}^{n} t/n$	$c_{av} = \sum_{i=1}^{n} c/n$

TABLE 6. - Determination of thickness-weighted average grade for a block

Workings (n), numbers	Thickness (t), ft	Grade (c), percent	Product, t x c
1..................................	t_1	c_1	$t_1 c_1$
2..................................	t_2	c_2	$t_2 c_2$
3..................................	t_3	c_3	$t_3 c_3$
. ...	.	.	.
. ...	.	.	.
. ...	.	.	.
N..................................	t_n	c_n	$t_n c_n$
Average (block A)...........	$\sum_{i=1}^{n} t_a$	$\sum_{i=1}^{n} c_a$	$\sum_{i=1}^{n} t_a c_a$

Note: Thickness-weighted average grade (c_{av}), percent: $c_{av} = \sum_{i=1}^{n} t_a c_a \Big/ \sum_{i=1}^{n} t_a$.

The arithmetic average procedure is the simplest and most rapid method of computation; accuracy depends on the quality, quantity, density, and distribution of observations; in turn these factors depend on the genetic type and size of the deposit.

The system is accurate in uniform deposits; accuracy decreases in nonuniform deposits, even if the distribution of observations is done by a regular pattern but with insufficient density. In complex deposits accuracy is greater in regularly distributed workings than in irregular. An important disadvantage of this system is the lack of quality and quantity distribution of valuable components in space.

TABLE 7. - Computation of reserves and average factors for the entire body

Block	Area (S), sq ft	Thickness (t), ft	Volume (V), cu ft	Weight factor (F), cu ft/ton	Raw material reserves (Q), tons	Valuable component: Average grade (c), percent	Valuable component: Reserves (P), tons
A..........	S_a	t_a	V_a	F	Q_a	c_a	P_a
B..........	S_b	t_b	V_b	F	Q_b	c_b	P_b
....	.	.	.	.	.	.	.
....	.	.	.	.	.	.	.
....	.	.	.	.	.	.	.
N..........	S_n	t_n	V_n	F	Q_n	c_n	P_n
Total...	$\sum_{i=1}^{n} S$		$\sum_{i=1}^{n} V$		$\sum_{i=1}^{n} Q$		$\sum_{i=1}^{n} P$
Average.		$t_{av} = \frac{\sum_{i=1}^{n} V}{\sum_{i=1}^{n} S}$				$c_{av} = \frac{\sum_{i=1}^{n} P}{\sum_{i=1}^{n} Q}$	

Statistical Factors

Factors for reserve computations of a certain mineral commodity are determined in terms of production or value yield on the basis of exploration, past mining experience, or smelter returns gathered for the same mineral body, or even similar ones. These factors are usually expressed as percent of component or value recovered per unit of area, volume, or weight; that is

	English system	Metric system
Placer deposits:		
Gold and precious metals.	Ounces per cubic yard or ton (short, long); cents per cubic yard, square foot or square yard; milligrams per cubic yard or ton.	Grams per cubic meter or metric ton.
Heavy minerals....	Pounds per cubic yard or ton..	Kilograms per cubic meter or metric ton.
Coal deposits.......	Tons per square foot, acre or section.	Metric tons per square meter or square kilometer.
Base metals and many nonmetallics.	Percent of weight or pounds per ton.	Percent of weight or kilograms per ton.

The use of statistical factors often may be the only practical way of computing potential resources for a mine or district. The accuracy depends on the geologic interpretation of the mineral deposit, as well as on computed factors.

Reserves of uniform bedded deposits, coal, phosphate rock, and clay have been computed by the method of analogy from the results of spot drill holes and exposures in trenches and other surface workings. Reserves of phosphate rock available for open pit mining in Idaho were computed by the Bureau of Mines on the basis of detailed geologic sections, sample analyses, geologic maps, and other published data. Inferred reserves of a syncline, for example, illustrated in figure 21 in part 1 of this report were based on sections measured about 1 mile from the area (table 8).

TABLE 8. - Reserve computations--method of analogy[1]

Block	Middle section (S), sq ft	Thickness accepted (t), ft	Volume (V), cu ft	Weight factor (F), cu ft/ton	Phosphate rock (Q), short tons	Grade (c), percent	P_2O_5, short tons
Block A, fig. 21, overturned section, 4,260 feet long							
Acid grade.........	715,000	5.0	3,575,000	11.3	316,400	34.2	108,200
Do............	715,000	9.9	7,078,000	11.3	626,400	32.5	203,600
Furnace grade......	715,000	6.5	4,648,000	12.0	387,300	26.9	104,200
Do............	715,000	15.0	10,725,000	12.0	893,700	28.1	251,100
Beneficiation grade	715,000	13.4	9,581,000	13.0	737,000	19.2	141,500
Total or average	715,000	49.8	35,607,000	-	2,960,800	27.3	808,600
Block B, fig. 21, normal dip, 2,050 feet long							
Acid grade.........	570,000	5.0	2,850,000	11.3	252,200	34.2	86,300
Do............	570,000	9.9	5,643,000	11.3	499,400	32.5	162,300
Furnace grade......	570,000	6.5	3,705,000	12.0	308,800	26.9	83,100
Do............	570,000	15.0	8,550,000	12.0	712,500	28.1	200,200
Beneficiation grade	570,000	13.4	7,638,000	13.0	587,500	19.2	112,800
Total or average	570,000	49.8	28,386,000	-	2,360,400	27.3	644,700

[1] Computations made for potential surface resources in a phosphorite deposit in Idaho. Section is composed of 2 acid grade zones (+31 percent P_2O_5), 2 furnace grade seams (24 to 31 percent P_2O_5), and the remainder beneficiation grade (18 to 24 percent P_2O_5). True bed thickness is 49.8 feet.

Method of Geologic Blocks

Although the method of geologic blocks has been widely used by earth scientists for many years, it was not until 1950 that its principles were first discussed and its name accepted (57); the procedure is also known as the method of analogy and general outline (63).

A geologic block may be the entire mineral deposit or a relatively small portion of it, outlined on a map by interpretation of exploratory data. The block sides may coincide with the natural boundaries of the deposit, or be delineated on the basis of geological features, structural deformations, or variations in thickness and grade. In addition, blocks also may be outlined on the basis of physiographic factors; adaptability to certain mining methods; availability of mineral raw material at depth; possibilities of utilization; requirements for beneficiation and processing; or property, section, township, or State boundaries.

The factors are determined from available exploratory data or may be adapted from results of spot sampling, production averaging, or data from other parts of the same deposit. Cutoff grade is determined by geologic and mining considerations and processing. Interpretation of data may be by the rules of gradual changes, nearest points, or generalization. The parameters of each geologic block and the entire

body are determined by procedure described for the method of analogy. The average grade of an individual block is computed either by the arithmetic average (table 5), weighted-average (table 6), or by statistical analysis. Reserves of each block are computed as the product of area and average factors; total reserves are the sum of all individual blocks (table 7).

Depending on the extent of the geologic knowledge of the deposit, all categories of mineral resources may be computed by the geologic blocks method. Accuracy of computations depends, essentially, upon the accuracy of factors accepted for each block and, to a lesser extent, on the accuracy of block area determinations. They may be as accurate as any other method, when a proper number of observations support the computation of factors for a certain deposit. On the other hand, the computations by this method may be speculative or purely academic, when the factors are based on an insufficient number and density of observations.

Examples of computations by this method are quite common in the early stages of exploration of bedded deposits; that is, phosphate rock, limestone, gypsum, and coal (fig. 30). It is often the only method that can be used when the deposit is irregular.

An excellent example of resource computations by geologic blocks has been published by the Geological Survey for uranium and vanadium deposits of the Colorado Plateau (8). The ore bodies are roughly tabular and generally paralleling the bedding of the sandstone host rock. They are irregular, often small in size (less than 5,000 tons), of variable thickness with uranium and vanadium values distributed erratically. Computations have been based on drill holes, underground openings, observations of natural outcrops, and production records. Often the number of observations for individual deposits were restricted.

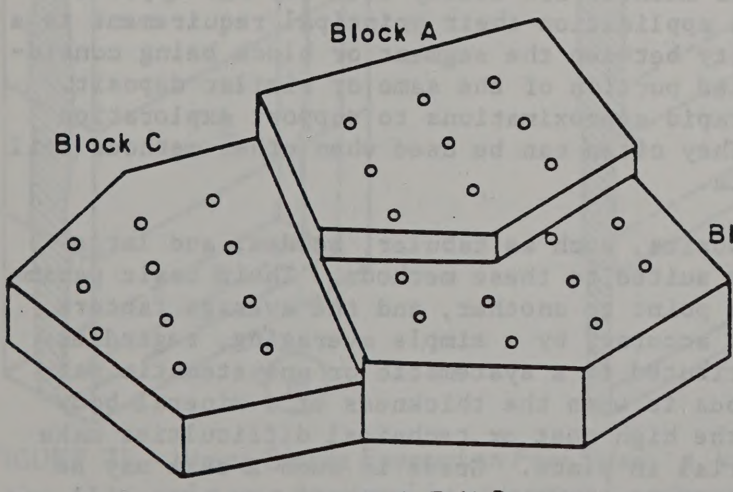


FIGURE 30. - Geologic Blocks Method.

At Lvov-Katin (near Moscow, U.S.S.R.) coal reserve computations were made by both geologic blocks and polygon methods (64). Geologic blocks were selected on the basis of bed thickness, as there were only small variations in coal quality (ash, sulfur, etc.). A total of 15 principal and 80 supplementary geologic blocks were used as compared with 260 blocks by the polygon method. The geologic blocks method revealed the presence of areas of varied and

sharply reduced coal thickness. As the geologic and mining conditions were different from other blocks, this area required additional exploration to permit reserve computations of the same category as the rest of the deposit.

Advantages

The average factors and area methods of reserve computations are relatively simple; their use, however, requires training and experience. Areas are measured by planimeter, computed, or scaled from maps. In general, the factors are determined by a minimum number of simple calculations; no special or detailed maps are needed. The procedure is flexible and requires no complex formulas; computations can be made for individual blocks, panels, levels, segments, or for the entire mineral body.

These methods are adaptable to all types of deposits and to all stages of development; they allow rapid and continuous evaluation of factual data, thus permitting improved engineering planning.

Individual observations of thickness and grade are often unconfirmed with respect to their localities; therefore, computations of average factors usually do not require area weighting. Changes in reserves of a mine, whether due to extraction or continuing exploration, can be easily made by subtracting or adding respective areas, or by determining new or corrected areas.

Accuracy of the computations varies depending on the type of deposit, number of blocks, and density of observations. When a deposit is quite uniform and average factors are computed on the basis of a sufficient number of observations, results are accurate.

Application

Analogous and geologic blocks methods are widely used for all types of mineral deposits. For successful application their principal requirement is a geologic and geochemical similarity between the segment or block being considered, and a more thoroughly studied portion of the same or similar deposit. Both methods are convenient for rapid approximations to support exploration and everyday mining decisions. They often can be used when other methods fail because of lack of sufficient data.

Certain types of mineral deposits, such as tabular, bedded, and large placer deposits, are particularly suited to these methods. Their basic parameters vary only slightly from one point to another, and the average factors may be determined with sufficient accuracy by a simple averaging, regardless of whether observations are distributed in a systematic or unsystematic pattern. Another use of these methods is when the thickness of a mineral body can be accurately measured, but the high cost or technical difficulties make it impossible to sample raw material in place. Grade in such a case may be computed from past production, other portions of the deposit, or even mill and smelter returns.

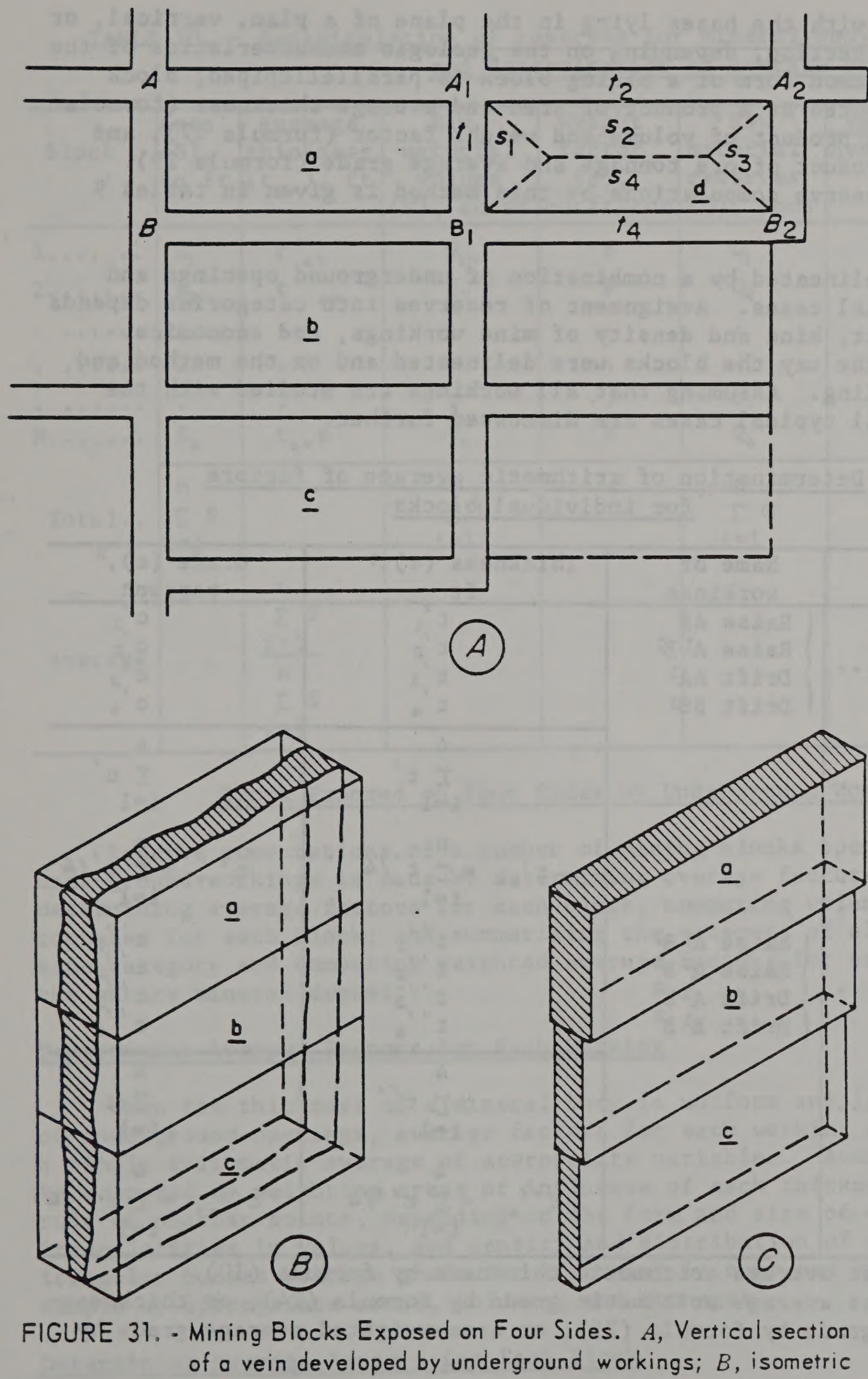


FIGURE 31. - Mining Blocks Exposed on Four Sides. *A*, Vertical section of a vein developed by underground workings; *B*, isometric drawing of three mining blocks *a*, *b*, and *c* (Note—vein thickness less than width of workings); *C*, geometric interpretation of the same blocks for computations.

Both methods should be used with discretion, because the accuracy for a deposit may depend on personal interpretation, rather than objective geologic observations and sampling.

Mining Blocks Method

The mining blocks is also known in the mineral industry as longitudinal section, mine extraction, and mine exploitation (11, 28, 51). A mining block may be defined as a portion of a mineral body delineated on four sides by workings, or bounded by workings on three or less sides, and by survey or arbitrary lines on the remaining sides (fig. 31). The size and form of the mining block is determined by exploration and development workings, geologic features, technical, and economic considerations.

In practice, mining blocks are generally

rectangular in shape with the bases lying in the plane of a plan, vertical, or incline longitudinal section, depending on the geologic characteristics of the deposit. The most common form of a mining block is parallellepiped; block volume of ore is computed as a product of area and average thickness (formula 15), ore tonnage as a product of volume and weight factor (formula 17), and metal tonnage as a product of ore tonnage and average grade (formula 29). The usual form for reserve computations by this method is given in tables 9 and 10.

Mining blocks delineated by a combination of underground openings and drill holes are special cases. Assignment of reserves into categories depends on the type of deposit, kind and density of mine workings, and economics. Accuracy depends on the way the blocks were delineated and on the method and accuracy of the sampling. Assuming that all workings are studied with the same accuracy, several typical cases are discussed further.

TABLE 9. - Determination of arithmetic average of factors for individual blocks

Block	Name of workings	Thickness (t),[1] ft	Grade (c),[2] percent
1....................	Raise AB Raise A^1B^1 Drift AA^1 Drift BB^1	t'_1 t'_2 t'_3 t'_4	c'_1 c'_2 c'_3 c'_4
Total............		$\sum_{i=1}^{4} t'$	$\sum_{i=1}^{4} c'$
Average..........		$t'_{av} = \sum_{i=1}^{n} t'/4$	$c'_{av} = \sum_{i=1}^{n} c'/4$
2....................	Raise A^1B^1 Raise A^2B^2 Drift A^1B^2 Drift B^1B^2	t''_1 t''_2 t''_3 t''_4	c''_1 c''_2 c''_3 c''_4
Total............		$\sum_{i=1}^{4} t''$	$\sum_{i=1}^{4} c''$
Average..........		$t''_{av} = \sum_{i=1}^{4} t''/4$	$c''_{av} = \sum_{i=1}^{4} c''/4$

[1](t) are determined as average arithmetic thickness by formula (10).
[2](c) are determined as average arithmetic grade by formula (24), or thickness-weighted average grade by formula (25), or area-weighted average grade by formula (26).

TABLE 10. - Recapitulation of reserves for mineral body (by categories) and determination of average grade

Block	Area (S), sq ft	Average thickness (t_{av}), ft	Raw material volume (V), cu ft	Weight factor (F), cu ft/ton	Raw material reserves (Q), tons	Valuable components: Average grade (c_{av}), percent	Valuable components: Reserves (P) tons
1.......	S_1	t'_{av}	V_1	F	Q_1	c'_{av}	P_1
2.......	S_2	t''_{av}	V_2	F	Q_2	c''_{av}	P_2
. ...	.	.	.	.	.	.	.
. ...	.	.	.	.	.	.	.
. ...	.	.	.	.	.	.	.
N.......	S_n	$t_{av}n$	V_n	F	Q_n	c^n_{av}	P_n
Total..	$\sum_{i=1}^{n} S$		$\sum_{i=1}^{n} V$		$\sum_{i=1}^{n} Q$		$\sum_{i=1}^{n} P$
Average		$\dfrac{\sum_{i=1}^{n} S}{\sum_{i=1}^{n} S}$				$\dfrac{\sum_{i=1}^{n} P}{\sum_{i=1}^{n} Q}$	

Block Exposed on Four Sides by Underground Workings

Reserve computations of a number of mining blocks opened on all sides by underground workings is made by determining average factors for each workings; determining average factors for each block; computing volume, ore, and metal tonnages for each block; and summarizing the reserves of all blocks of the same category and computing weighted average factors for each category and for the entire mineral deposit.

Determining Average Factors for Each Working

When the thickness of a mineral body is uniform and less than the width of underground openings, average factors for each working are usually found by a simple arithmetic average of appropriate variables. Average thickness may be computed by weighting areas of influence of each thickness according to the rule of nearest points, depending on the form and size of the mineral body, irregularities in values, and density and distribution of observations. In irregular bodies average grade for a working is computed by weighting each sample by appropriate areas, volumes, and tonnages.

Determining Average Factors for Each Block

When the thickness of mineral body is less than the width of the opening, average factors are computed as follows:

lengths of all sides are equal,

$$t_{av} = \frac{t_1 + t_2 + t_3 + t_4}{4} \tag{33}$$

$$c_{av} = \frac{c_1 + c_2 + c_3 + c_4}{4}, \tag{34}$$

where t_1, t_2, t_3, and t_4 are thicknesses measured or computed for each working; c_1, c_2, c_3, anc c_4 are grades for the same workings.

When lengths of sides are unequal and there is no relationship between thickness and grade, average factors of a mining block may be computed by weighting each working according to its length, L_1, L_2, L_3, and L_4,

$$t_{av} = \frac{t_1 L_1 + t_2 L_2 + t_3 L_3 + t_4 L_4}{L_1 + L_2 + L_3 + L_4} \tag{35}$$

and

$$c_{av} = \frac{c_1 L_1 + c_2 L_2 + c_3 L_3 + c_4 L_4}{L_1 + L_2 + L_3 + L_4}. \tag{36}$$

When lengths of sides are unequal and thicknesses and grades of workings vary considerably (fig. 31A), average factors are

$$t_{av} = \frac{t_1 s_1 + t_2 s_2 + t_3 s_3 + t_4 s_4}{s_1 + s_2 + s_3 + s_4} \tag{37}$$

and

$$c_{av} = \text{formula 26},$$

where s_1, s_2, s_3, and s_4 are areas of influences of each working found by the rule of nearest points.

When the ore thickness is more than the width of the mine openings and the blocks are developed by crosscuts on two levels, reserves may be computed from the crosscut data, as discussed further for a block exposed on two sides. Drift and raise samples between crosscuts are not required, if there are evidences of good ore continuity and grade uniformity. In the case of an irregular body the block may be divided into areas of influence to check results of original computations.

Block Exposed on Three Sides by Underground Workings

Average factors for a mining block exposed on three sides by underground workings are computed similar to the previous cases: (a) as an arithmetic average of three sides (disregarding length of workings); (b) by weighting variables of each side of the block according to the length of each working; (c) by computing first the factors for the fourth side from end-samples a and b of the existing sides (fig. 32A left) and, then, averaging the variables of all sides (this, however, increases the importance of the end-samples in

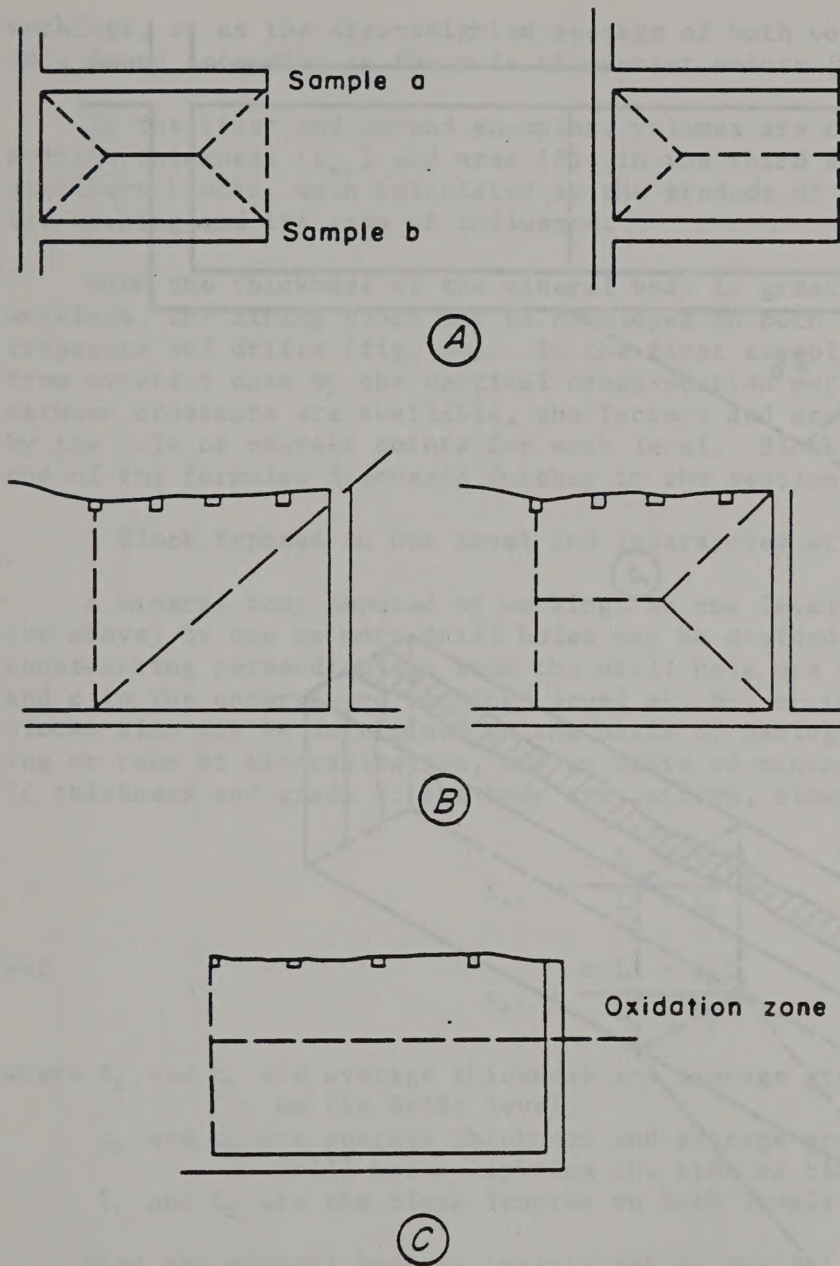


FIGURE 32. - Mining Blocks Exposed on Three Sides. *A*, Block exposed by two drifts and a raise; *B*, block exposed by adit, raise, and surface workings, no change in geology with depth; *C*, block exposed by adit, raise, and trenches, natural changes in thickness and grade with depth due to oxidation.

comparison with others); and (d) by weighting the areas of influence of each of the three existing workings (fig. 32A right).

A block delineated by an adit, raise, and surface workings, that is trenches or pits, is a special case and may be subdivided as above (fig. 32B right), or on the basis of accuracy of computations (category) (fig. 32B left). It may also be subdivided into blocks based on geologic evidences, such as degree of ore alteration, thickness, grade, zoning, or number of observations (32C).

Block Exposed on Two Sides

Underground Workings on Two Levels

When a mining block is developed on two levels (fig. 33, block 1) average factors for each level are found first. In uniform bodies the block factors are computed as the average of both levels; otherwise they are computed as area-weighted averages of both levels.

Intersecting Underground Workings

A mining block developed on two sides by underground intersecting workings; that is drift and raise, is a triangular prism (fig. 33, block 2). Block factors may be determined as the average of both workings, as the length-weighted average of both

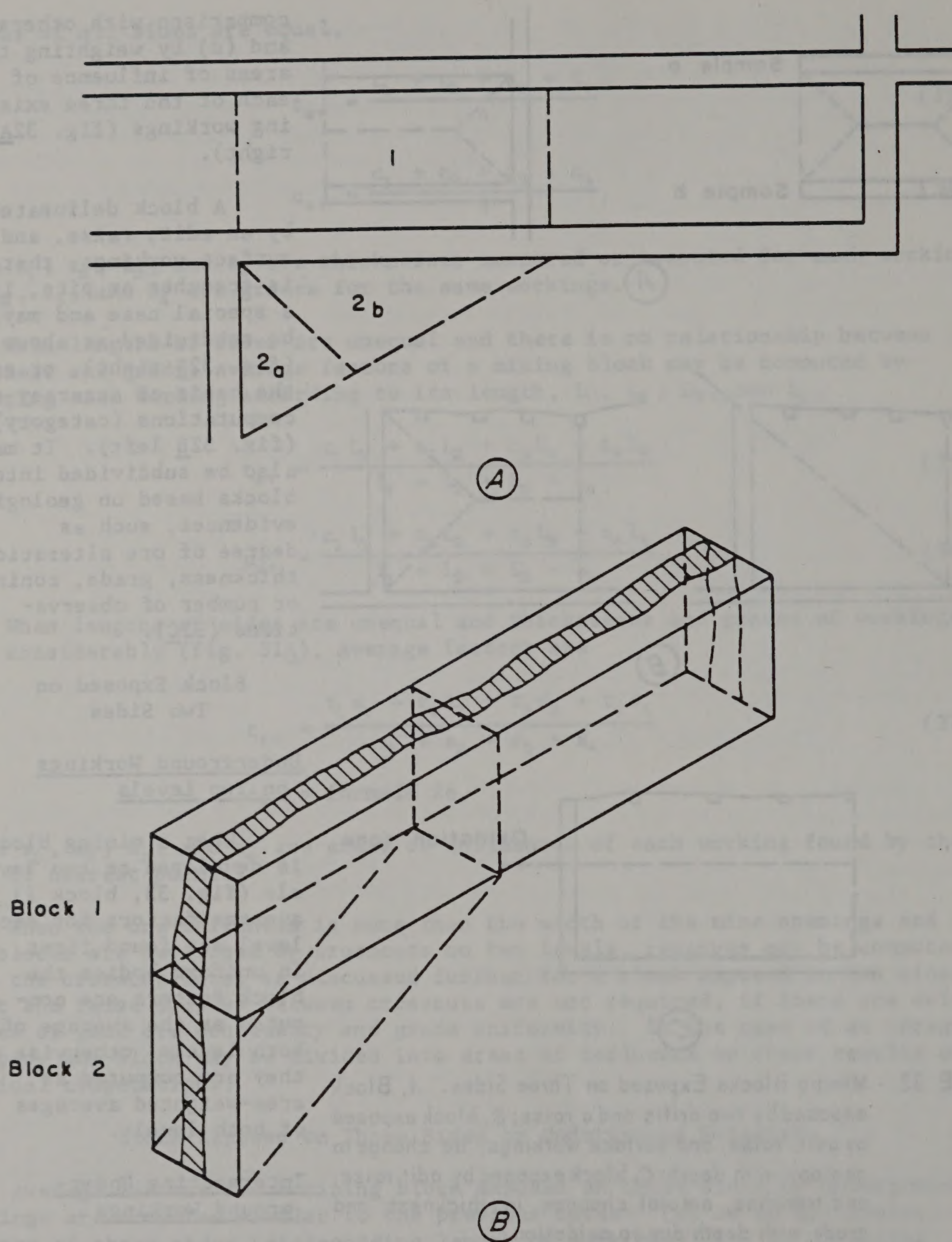


FIGURE 33. - Mining Blocks Exposed on Two Sides (Vein Thickness Less Than the Width of Workings). *A*, Blocks exposed between two parallel workings (1) and between two intersecting workings (2); *B*, isometric drawing of the same mining blocks.

workings, or as the area-weighted average of both workings with areas of influence found according to the rule of nearest points (blocks 2a and 2b).

In the first and second examples, volumes are computed as the product of average thickness (t_{av}) and area (S); in the third example as a sum of two auxiliary blocks, each calculated as the product of the average thickness of the working and its area of influence.

When the thickness of the mineral body is greater than the width of the workings, the mining block may be developed on both levels by crosscuts or by crosscuts and drifts (fig. 34). In the first example reserves are computed from crosscut data by the vertical cross-section methods. When drift samples between crosscuts are available, the factors and area reserves may be computed by the rule of nearest points for each level. Block reserves are computed by one of the formulas discussed further in the section on cross-section methods.

Block Exposed on One Level and Intersected at Depth by Drilling

A mineral body exposed by workings on one level and intersected at depth (or above) by one or more drill holes may be divided into mining blocks by constructing perpendiculars from the drill hole ore intersection level a, b, and c to the underground workings level a^1, b^1, c^1 (fig. 35A). Boundaries of blocks also may be determined on the basis of geologic criteria, such as zoning or rake of mineralization, and on basis of mining design and economics. If thickness and grade of the body are uniform, block factors are computed by

$$t_{av} = \frac{t_1 L_1 + t_2 L_2}{L_1 + L_2} \tag{38}$$

and

$$c_{av} = \frac{c_1 L_1 + c_2 L_2}{L_1 + L_2}, \tag{39}$$

where t_1 and c_1 are average thickness and average grade of ore in each block on the drift level;

t_2 and c_2 are average thickness and average grade of each two adjoining drill holes limiting the side of block; and

L_1 and L_2 are the block lengths on both levels.

When the mineral body is intersected by one drill hole, the mining block may be divided according to the rule of nearest points into two auxiliary blocks of varying accuracy and, therefore, of different categories (fig. 35B).

The average factors may be determined by formulas (57, p. 219),

$$t_{av} = \frac{3t_1 + t_2}{4} \tag{40}$$

and

$$c_{av} = \frac{3c_1 + c_2}{4}, \tag{41}$$

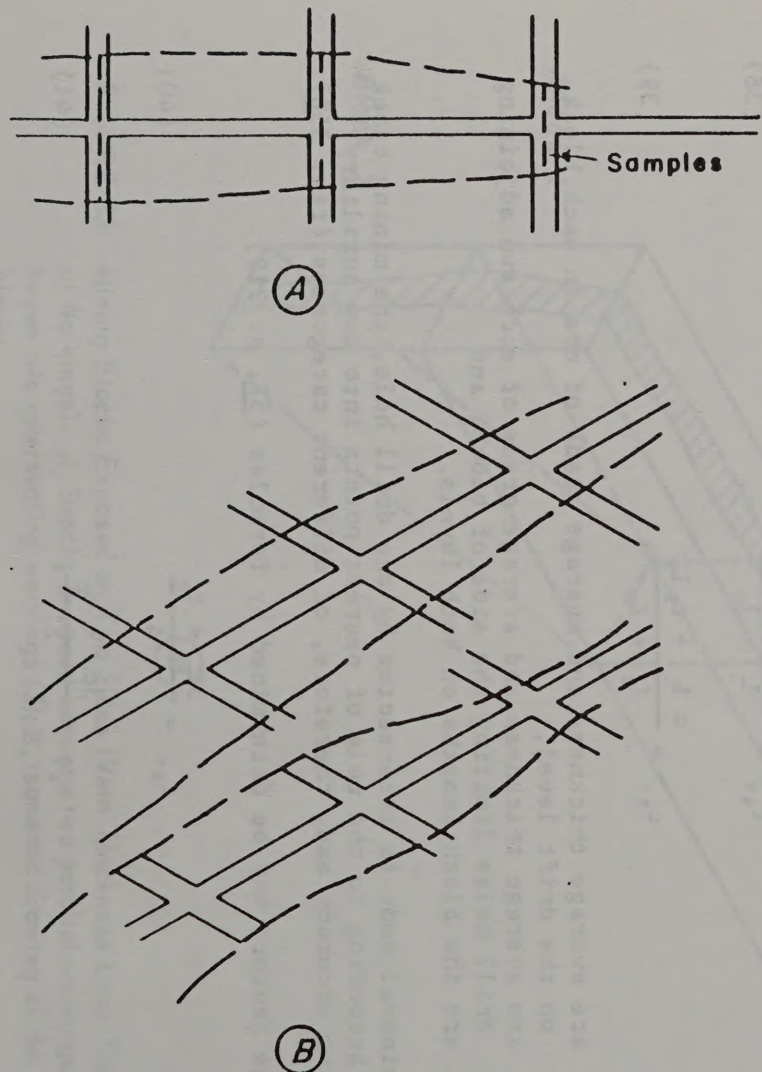


FIGURE 34. - Mining Blocks Exposed on Two Sides (Vein Thickness More Than the Width of Workings). *A*, Plan of two mining blocks delineated by crosscuts; *B*, block diagram of two mining blocks delineated on two levels by crosscuts.

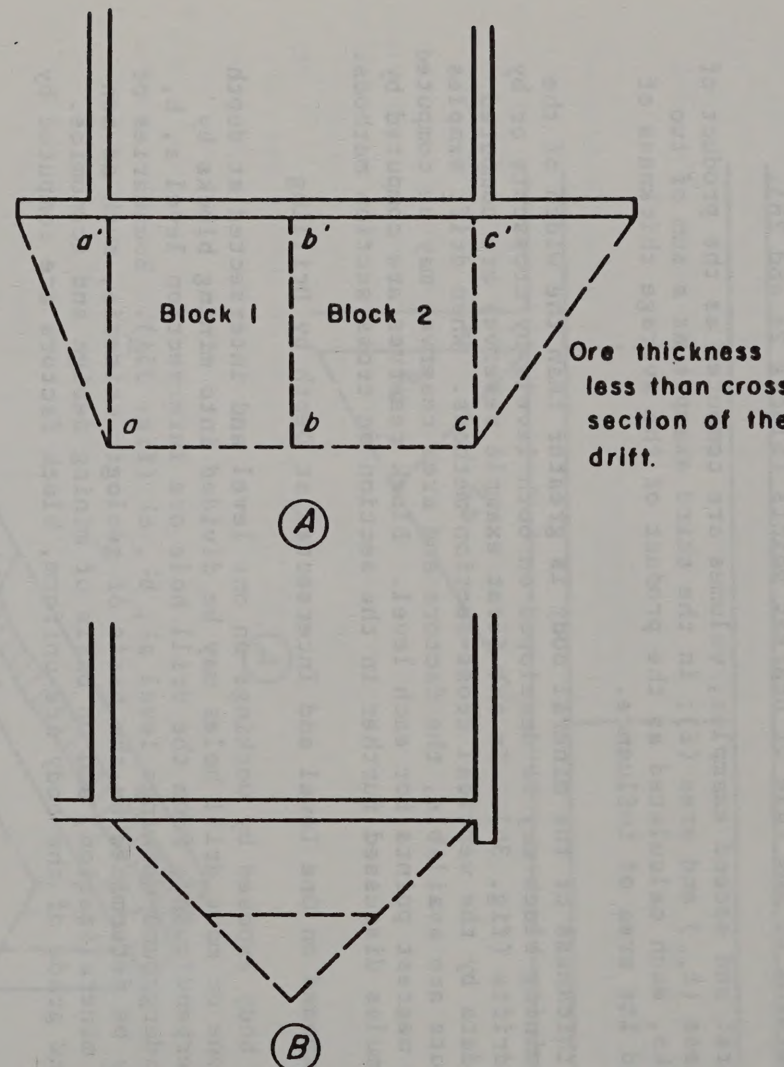


FIGURE 35. - Mining Blocks Exposed by Drift and Drill Holes. *A*, Block exposed on one level by a drift and intersected at depth by two or more drill holes; *B*, block exposed on one level by a drift and intersected at depth by one drill hole.

where t_1 and c_1 are average thickness and assay for the drift and t_2 and c_2 are thickness and assay for the drill hole.

When a large number of samples are available from the drift, drill hole information often may serve only to delineate the block and evaluate the mineralization. If each observation, whether it is from the drift or drill hole, is considered on an equal basis, then the computations of average thickness and grade are made by formulas.

In the example of numerous holes the formulas are

$$t_{av} = \frac{t_1 + t_2 + \ldots + t_n + t^1{}_1 + t^1{}_2 + \ldots + t^1{}_m}{n + m} \tag{42}$$

and

$$c_{av} = \frac{c_1 + c_2 + \ldots + c_n + c^1{}_1 + c^1{}_2 + \ldots + c^1{}_m}{n + m}. \tag{43}$$

With one hole they are

$$t_{av} = \frac{t_1 + t_2 + \ldots + t_n + t^1{}_1}{n + 1} \tag{44}$$

and

$$c_{av} = \frac{c_1 + c_2 + \ldots + c_n + c^1{}_1}{n + 1}. \tag{45}$$

In these computations,

t_1, t_2, ... t_n are thicknesses observed in the drift,
c_1, c_2, ... c_n are corresponding assays,
$t^1{}_1$, $t^1{}_2$, ... $t^1{}_m$ thicknesses observed in drill holes,
$c^1{}_1$, $c^1{}_2$, ... $c^1 m$ corresponding assays,
n - number of samples in the drift, and
m - number of drill holes.

Application

In order to apply the mining blocks method it is necessary to develop the mineral body into blocks (for extraction) by a sufficient distribution of workings. In general, reserves for uniform bodies are of the highest category; that is, proved or semiproved. Computations are relatively simple. Block reserves may be classified for mining purposes according to thickness, grade, and extraction cost. Thus, the method allows the operator to control the quality and the cost of production.

The method is flexible and may be used in all types of mineral deposits. The degree of error depends, to a great extent, on the genetic type of the deposit, on the density of workings, and on the distribution of observations. It is naturally adapted to sedimentary beds such as coal, to vertical and steeply dipping veins of thin and medium thickness, and to thin-bedded tabular

ore bodies, where grade and thickness undergo gradual changes and where mining and geologic features are similar to blocks already extracted. When weighted averaging of thickness and grade can be eliminated from the computations, it becomes a simple operation.

In nestlike, broken, or interrupted bodies, or where mineral values are distributed erratically, the relative error of this method may be excessive.

Cross-Section Methods

Principles and Requirements

The initial step in the application of cross-section methods is to divide the mineral body into blocks by constructing geologic sections at intervals along the transverse lines or at different levels in conformity with exploration workings, purpose of computations, and the nature of the deposit (figs. 36 and 37). The interval between the sections may be constant or may vary to suit the geology and mining requirements. When the intervals are unequal, formulas for computations are slightly more complicated.

Depending on the manner of the block construction there are three modifications of cross-section methods:

1. Standard method based on the rule of gradual changes (fig. 37A). Each internal block is confined by two sections and by an irregular lateral surface, and each end block by a single section and by an uneven lateral

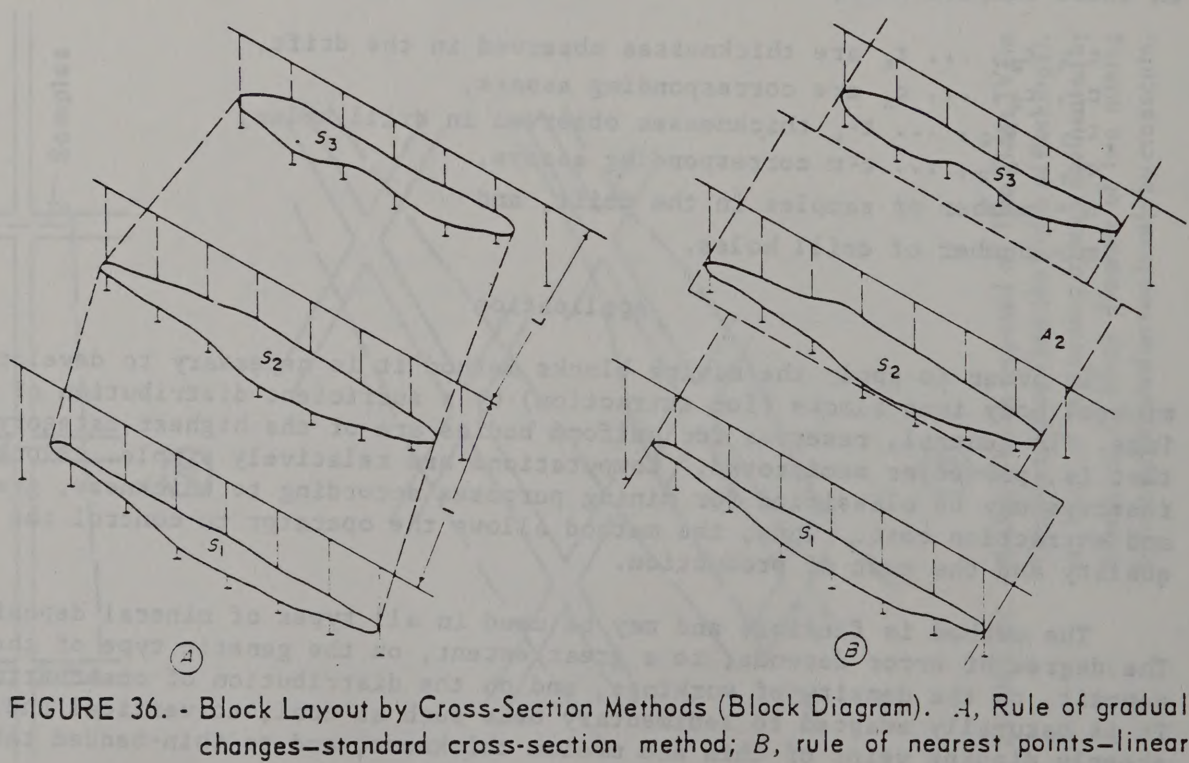


FIGURE 36. - Block Layout by Cross-Section Methods (Block Diagram). *A*, Rule of gradual changes—standard cross-section method; *B*, rule of nearest points—linear cross-section method.

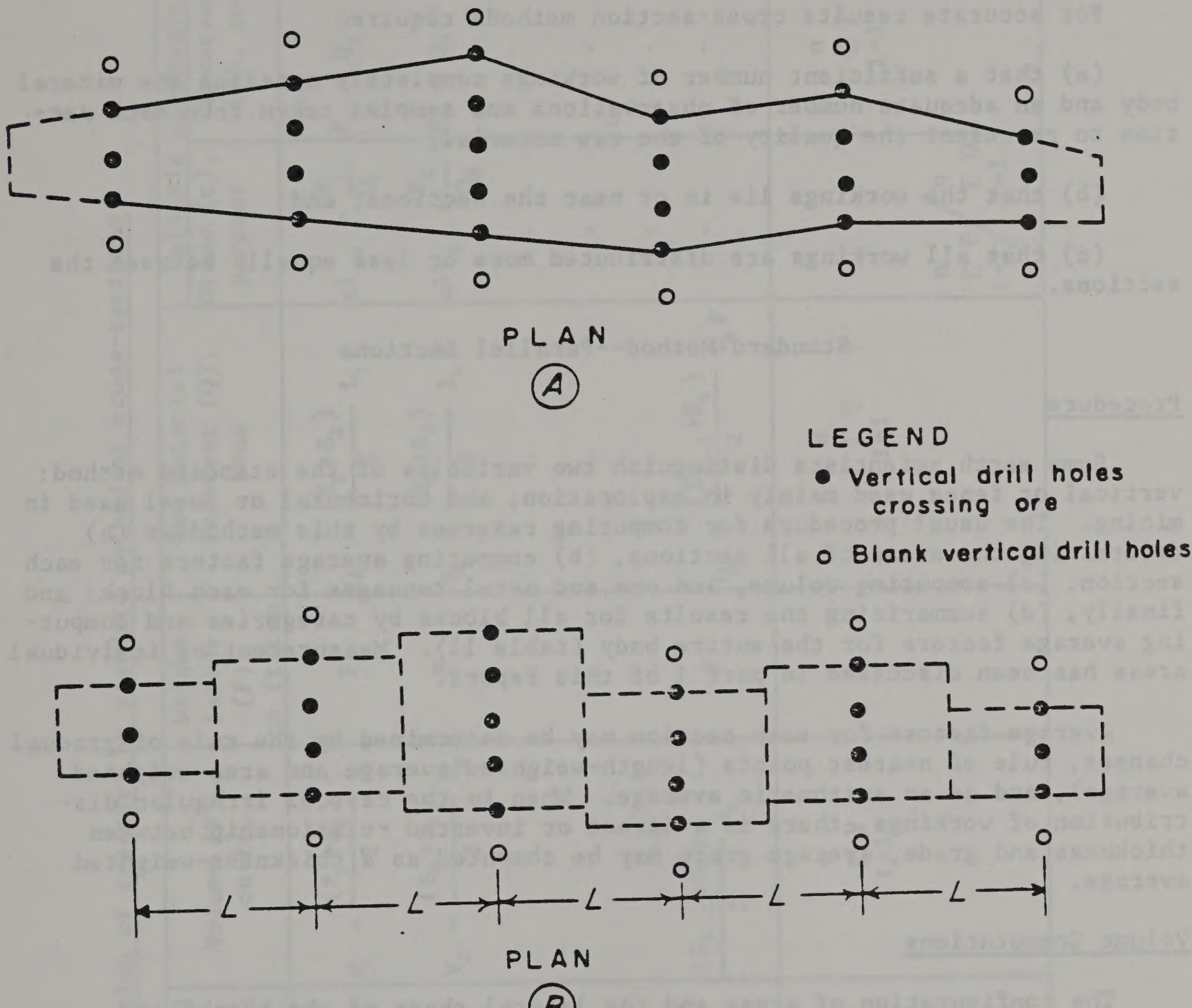


FIGURE 37. - Cross-Section Methods—Standard and Linear. *A*, Laying out blocks according to the rule of gradual changes; *B*, laying out blocks according to the rule of nearest points.

surface. Sections may be parallel or nonparallel, vertical, horizontal, or inclined;

2. Linear method based on the rule of nearest points (fig. 37B). Each block is defined by a section and a length equal to one-half the distance to the adjoining sections; and

3. Method of isolines based on the rule of gradual changes (see section entitled Method of Isolines).

For accurate results cross-section methods require

(a) that a sufficient number of workings completely crossing the mineral body and an adequate number of observations and samples taken from each section to represent the quality of the raw material;

(b) that the workings lie in or near the sections; and

(c) that all workings are distributed more or less equally between the sections.

Standard Method--Parallel Sections

Procedure

Some earth scientists distinguish two variables of the standard method: vertical or fence used mainly in exploration; and horizontal or level used in mining. The usual procedure for computing reserves by this method is (a) determining the areas of all sections, (b) computing average factors for each section; (c) computing volume, and ore and metal tonnages for each block; and finally, (d) summarizing the results for all blocks by categories and computing average factors for the entire body (table 11). Measurement of individual areas has been discussed in part 1 of this report.

Average factors for each section may be determined by the rule of gradual changes, rule of nearest points (length-weighted average and area-weighted average), and as an arithmetic average. When in the case of irregular distribution of workings, there is a direct or inverted relationship between thickness and grade, average grade may be computed as a thickness-weighted average.

Volume Computations

The configuration of areas and the lateral shape of the blocks are usually irregular and, in order to compute volume by solid geometry, areas are considered to be of equal size circles or polygonal figures; lateral surfaces of the blocks are disregarded.

Mean-Area Formula. - The simplest formula for a volume between two parallel sections with areas S_1 and S_2 and a perpendicular distance, L, between them is

$$V = \frac{(S_1 + S_2)}{2} L. \qquad (46)$$

This mean-area formula is precise when both areas are nearly similar in size and shape.

TABLE 11. - Computation of reserves by standard method of cross-sections

Blocks	Sections	Area (S), sq ft	Interval between sections (L), ft	Volume (V), cu ft	Weight factor (F), cu ft/ton	Raw material reserves (Q), tons	Valuable component: Grade (c), percent	Valuable component: Reserves (P), tons
1...........	{ A-A$_1$ B-B	S_1 S_2	} L_1	$V_1 = \frac{(S_1 + S_2)}{2} L_1$	F	$Q_1 = \frac{(S_1 + S_2)}{2} L_1 F$	$c^1{}_{av} = \frac{P_1}{Q_1}$	$P_1 = Q_1 c^1{}_{av}$
2...........	{ B-B$_1$ C-C	S_2 S_3	} L_2	$V_2 = \frac{(S_2 + S_3)}{2} L_2$	F	$Q_2 = \frac{(S_2 + S_3)}{2} L_2 F$	$c^{11}{}_{av} = \frac{P_2}{Q_2}$	$P_2 = Q_2 c^{11}{}_{av}$
. ...	.	.	.	.	.	.	.	.
. ...	.	.	.	.	.	.	.	.
. ...	.	.	.	.	.	.	.	.
N...........	-	S_{n-1} S_n	} L_n	$\frac{(S_{n-1} + S_n)}{2} L_n$	F	$Q_n = \frac{(S_{n-1} + S_n)}{2} L_n F$	-	-
Total....		$\sum_{i=1}^{n} S$	$\sum_{i=1}^{n} L$	$\sum_{i=1}^{n} V$		$\sum_{i=1}^{n} Q$		$\sum_{i=1}^{n} P$
Average..							$\sum_{i=1}^{n} P \Big/ \sum_{i=1}^{n} Q$	

A solid mineral body that has been divided into blocks by a series of evenly spaced parallel sections is computed by the "end-area" formula derived from formula (46),

$$V = (S_1 + 2S_2 + 2S_3 + \ldots + S_n)\,\frac{L}{2}, \tag{47}$$

where L equals the distance between sections. If the sections are unevenly spaced the formula for the volume of the entire body will be

$$V = \frac{(S_1 + S_2)}{2} L_1 + \frac{(S_2 + S_3)}{2} L_2 + \ldots + \frac{(S_{n-1} + S_n)}{2} L_n, \tag{48}$$

where L_1, L_2, L_3, ... L_n are perpendicular distances between the adjoining sections with areas S_1, S_2, S_3 ... S_n.

Wedge and Cone (Pyramid) Formulas. - End blocks of lenslike mineral bodies may be converted to a wedge or cone (pyramid) with the larger areas S in one section, tapering to a line or a point in the adjoining section (fig. 38A). If the block tapers to a line, volume is computed by wedge formula,

$$V = \frac{S}{2} L. \tag{49}$$

This formula, however, is precise only when the base is rectangular and the lateral faces are isosceles triangles and trapezoids. A more precise formula for the wedge is (42)

$$V = \frac{L}{6} (2a + a_1)\, b \sin \alpha, \tag{50}$$

where a and b are the lengths of sides of the base α - angle between a and b, and a_1 is the larger side of the trapezoid (38A right).

If the block tapers to a point (fig. 38B), volume is computed by cone formula,

$$V = \frac{S}{3} L. \tag{51}$$

Volume computed by the wedge formula is 50 percent larger than volume computed by the cone formula.

Frustum Formula. - When S_1 and S_2 vary in size, but are similar (fig. 38C), frustum of a cone or pyramid formula is used to compute the block volume,

$$V = \frac{L}{3} (S_1 + S_2 + \sqrt{S_1 S_2}). \tag{52}$$

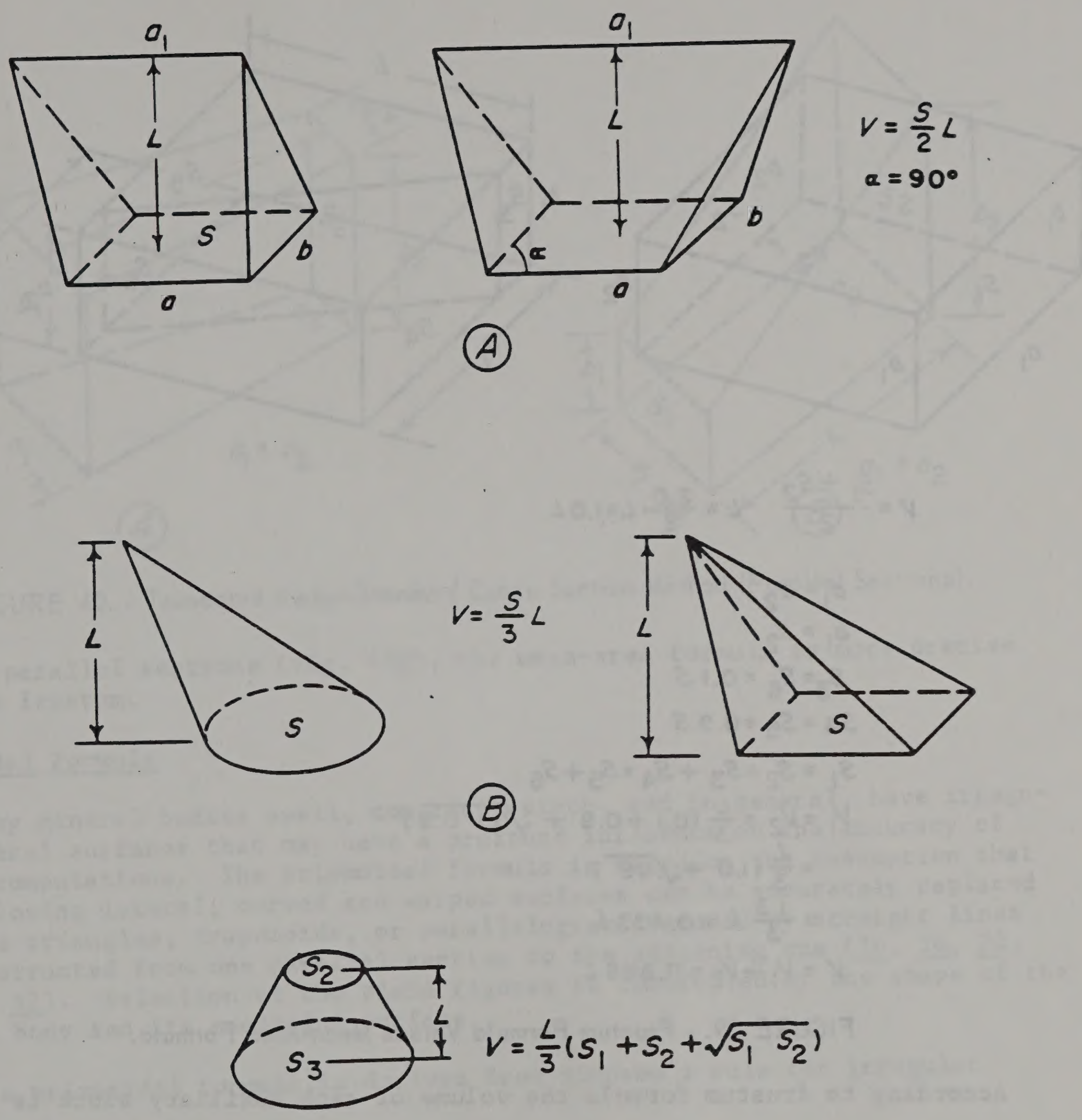
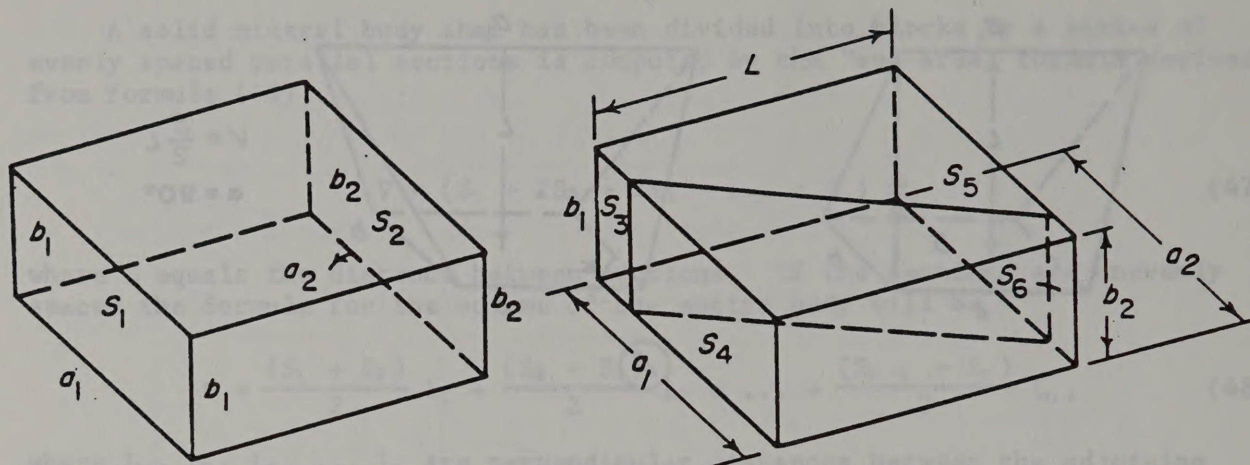


FIGURE 38. - Standard Cross-Section Method (Parallel Sections). *A*, Common wedge formula; *B*, cone (pyramid) formula; *C*, frustum of a cone formula.

In practice, the frustum formula is avoided because of complications involved in computing square roots and in certain cases it is less accurate than the prism formula. Let us consider a block of a regular prism (volume of which is V = S x L) and divide it into two auxiliary prisms in the form of truncated wedges with areas S_4 and S_5 equal to 0.9S and areas S_3 and S_6 equal to 0.1S (fig. 39) (30).



$$V = \frac{S_1 + S_2}{2} \quad L = \frac{2.0}{2} L = 1.0 L$$

$$a_1 = a_2$$
$$b_1 = b_2$$
$$S_3 = S_6 = 0.1\, S$$
$$S4 = S_5 = 0.9\, S$$
$$S_1 = S_2 = S_3 + S_4 = S_5 + S_6$$
$$V_1 = V_2 = \frac{L}{3}(0.1 + 0.9 + \sqrt{0.1\ 0.9})$$
$$= \frac{L}{3}(1.0 + \sqrt{.09})$$
$$= \frac{1.3}{3} L = 0.433 L$$
$$V = V_1 + V_2 = 0.866 L$$

FIGURE 39. - Frustum Formula Versus Mean-Area Formula.

According to frustum formula the volume of each auxiliary block is

$$V_1 = V_2 = \frac{L}{3}\,(0.1 + 0.9 + \sqrt{(0.1)\ (0.9)}) = 0.433\ L,$$

and the volume of the regular prism is

$$V = V_1 + V_2 \doteq 0.866\ L.$$

In this example the total block volume, by frustum formula, is 13.4 percent less than the volume computed by the regular prism formula. It is concluded, therefore, that the frustum formula is inaccurate in wedgelike bodies. Thus, when the areas delineating the truncated wedge blocks have equal sides, such as heights b_1 and b_2 between two levels (fig. 40<u>A</u>), or thicknesses a_1 and a_2

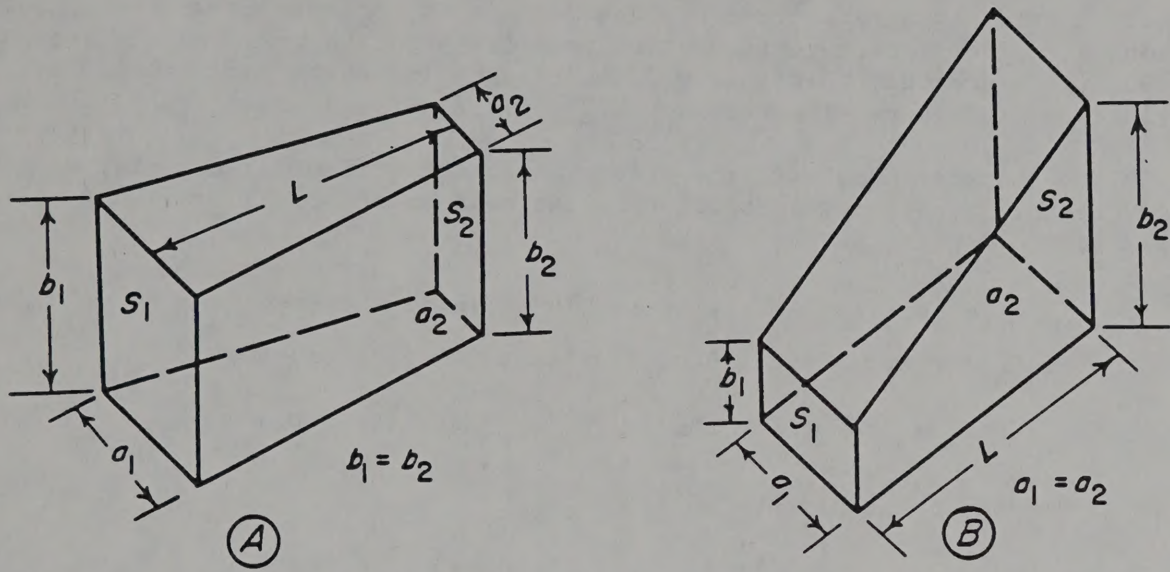


FIGURE 40. - Truncated Wedge-Standard Cross-Section Method (Parallel Sections).

between parallel sections (fig. 40B), the mean-area formula is more precise than the frustum.

Prismoidal Formula

Many mineral bodies swell, contract, pinch, and in general, have irregular lateral surfaces that may have a profound influence on the accuracy of volume computations. The prismoidal formula is based on the assumption that the enclosing lateral, curved and warped surfaces can be accurately replaced by plane triangles, trapezoids, or parallelograms bounded by straight lines and constructed from one parallel section to the adjoining one (10, 14, 20, 40, 42, 62). Selection of the plane figures is controlled by the shape of the mineral body and its enclosing surface.

The prismoidal formula is derived from Simpson's rule for irregular areas.

$$V = (S_1 + 4M + S_2)\,\frac{L}{6}, \tag{53}$$

where M is the area of an auxiliary plane section parallel to and midway between sections S_1 and S_2. Construction of this auxiliary section is based on interpolation of longitudinal and cross sections and by interpretation of the geology of the mineral body (39, p. 581, 28, p. 117). Only in exceptional cases is M an average of S_1 and S_2. Auxiliary area constructions require additional work.

This formula is advantageous when a mineral body is divided into blocks by a series of closely spaced cross sections. Alternate or odd number sections may be regarded as end block sections, and each intervening or even

numbered section as a midsection M. The formula is recommended when the cross sections are of different configuration and more accurate computations are desired; it is commonly used in civil engineering for earth work and is described in field surveying handbooks (39, p. 582, 57, v. 1, p. 231).

Volume computations, with the precision of the prismoidal formula, may be made by reducing the results computed by the mean-area formula by a "prismoidal correction factor",

$$V_{prismoidal} = V_{mean\text{-}area} - C_{prismoidal\ correction},$$

where C, the factor for any triangular prismatoid, is equal to

$$C = \frac{L}{3}(S_1 - 2M + S_2) = \frac{L}{12}(a_1 - a_2)(b_1 - b_2) \tag{54}$$

and is expressed in cubic feet.

$$M = \frac{(a_1 + a_2)}{2}\frac{(b_1 + b_2)}{2}, \tag{55}$$

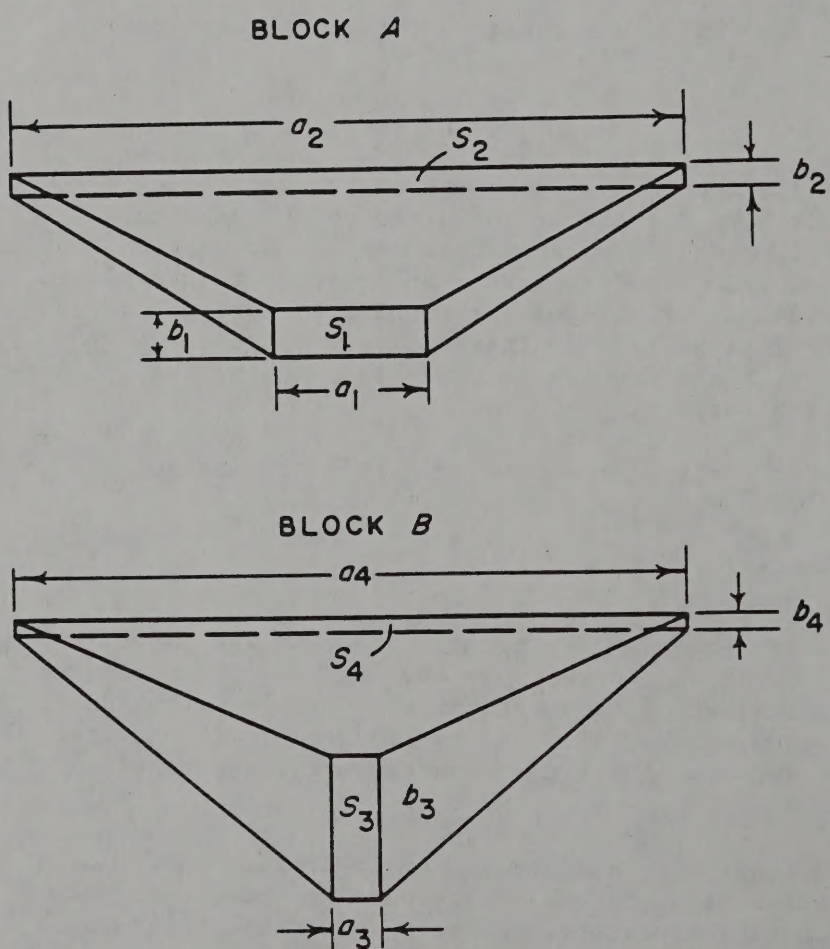


FIGURE 41.-Blocks Between Parallel Sections. Influence of the shape of areas on volume computations.

where a_1 and b_1 are sides of area S_1 and a_2 and b_2 are sides of area S_2 (fig. 41, Block A). Tables of values for triangular prisms for a distance of 100 feet between sections, and for prismoidal corrections are available in publications related to railway and other earth excavation problems. The obelisk formula, described in some publications as a separate formula (30), is a modification of a prismoidal formula derived by substituting equation (55) for M.

$$V = \frac{L}{6}(S_1 + 4M + S_2) = \frac{L}{6}\left[S_1 + 4\,\frac{(a_1 + a_2)\,(b_1 + b_2)}{4} + S_2\right]$$

$$= \frac{L}{3}\left[S_1 + S_2 + \frac{(a_1 b_2 + a_2 b_1)}{2}\right]. \qquad (56)$$

The influence of the shape of parallel areas on volume computations is well demonstrated by the following. Comparing the results of computations by mean-area, frustum, and prismoidal formulas for two bodies with the same value base areas S_1, S_2, S_3, and S_4, but of different shape (fig. 41), gives

	Block A	Block B
Distance between sections........feet..	15	15
Dimensions of bodies, feet:		
a_1	16	-
b_1	5	-
S_1	80	-
a_2	40	-
b_2	2	-
S_2	80	-
a_3	-	5
b_3	-	16
S_3	-	80
a_4	-	40
b_4	-	2
S_4	-	80
Mean-area..........................ft^3..	1,200	1,200
Frustum............................ft^3..	1,200	1,200
Prismoidal.........................ft^3..	1,380	2,425
Relative errors of mean-area and frustum formulas............percent..	13.0	50.5

Thus, in our example the volumes of blocks A and B computed by the mean-area and frustum formulas are 13.0 and 50.5 percent less than the values computed by prismoidal formula.

If values b_1 and b_2 are equal, the prismoidal (obelisk) formula converts to the mean-area formula.

$$V = \frac{L}{3}\left[S_1 + S_2 + \frac{(a_1 b_2 + a_2 b_1)}{2}\right] = \frac{L}{3}\left[S_1 + S_2 + \frac{(S_1 + S_2)}{2}\right] = L\,\frac{(S_1 + S_2)}{2}$$

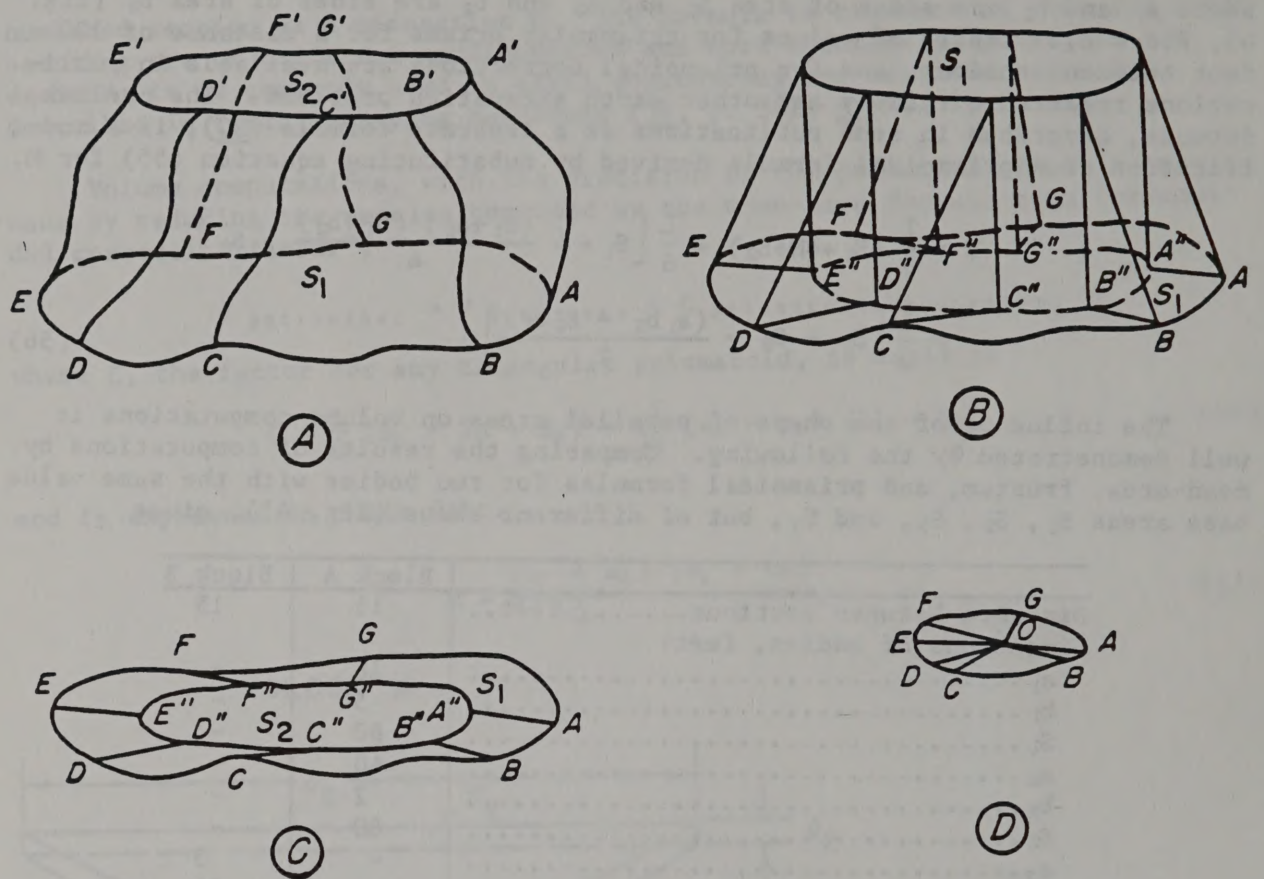


FIGURE 42. - Construction of Auxiliary Area R for Bauman's Formula. *A*, Block between parallel sections with irregular lateral surface; *B*, same block with linear lateral surface and construction of generators; *C*, intermediate drawing showing how to find projections of generators; *D*, construction of auxiliary area R.

Thus, when any pair of side values of a and b are equal or nearly equal, the mean-area formula is accurate. This is in general a common case of volume computations, such as a block having equal height or thickness.

The prismoidal formula has been used in computations of complex vein ore bodies in the Coeur d'Alene mining district, Idaho (10).

Bauman's Formula. - Among the less common formulas for volume computations is one offered by Bauman (2, 31). This graphic formula has limited use.

When the lateral and irregular surface of a mineral body between two parallel sections is assumed to be linear (fig. 42), the volume of the block may be computed by

$$V = \frac{L}{2}(S_1 + S_2) - \frac{LR}{6} \text{ or } V = \frac{L}{6}(3S_1 + 3S_2 - R), \qquad (57)$$

where S_1 and S_2 are areas of cross sections,
L is distance between sections, and
R is an auxiliary area, graphically constructed as follows:

1. Draw both end areas and construct projections (AA^{11}, BB^{11}, ...) from the generators (AA^{1}, BB^{1}, ...) of the lateral surface on the same plane (figs. 42B and 42C).
2. From point 0 (fig. 42D) construct lines of equal length and direction as the lines AA^{11}, B^{11}, ... If the generators are taken in sufficient quantity and all points A, B, C, ... are connected by a curved line, the resultant figure will be the auxiliary area R.

Construction of auxiliary areas requires experience and time, therefore, limiting the use of the formula. The common prism and pyramid formulas may be derived from Bauman's formula by making appropriate assumptions. When area S_1 is equal to S_2, the auxiliary area R is zero and Bauman's formula is the same as a prism or cylinder. When area S_2 is zero, the auxiliary area R will be equal to S_1, and Bauman's formula is that of a pyramid.

Tonnage Computations

The product of block volume and weight factors produces the tonnage of raw material; the product of the latter and average grade equals the reserves of valuable component. Another manner of tonnage computation for each block consists of determining "section reserves" for a slice of one unit in width, and computing block reserves as the product of half the sum of the section reserves and the block length.

Section reserves are computed as the total of the reserves between workings and may be determined by the rule of gradual changes (fig. 43A) or the rule of nearest points (fig. 43C). Section reserves are often termed linear reserves when taken along exploration lines. The term "linear reserves", in this report, is set apart for reserves computed for one square unit of area; that is, square foot, square yard, etc. (fig. 43B), and the term "area reserves" is used for areas of substantial size such as acre and square mile.

The Standard Method for Nonparallel Sections

Sections constructed along exploration lines may converge or diverge because of changes in the strike of the mineral body. The angles between sections and exploration lines may range from oblique to obtuse, depending on strike variations. Formulas offered for computing reserves with nonparallel sections are discussed in the following sections.

Angle Less Than 10 Degrees

When the angle of intersection is less than 10 degrees, Zolotarev offered the formula (fig. 44) (27, 31, 48, 57, 66),

$$V = \frac{(S_1 + S_2)}{2}\frac{(h_1 + h_2)}{2}, \tag{58}$$

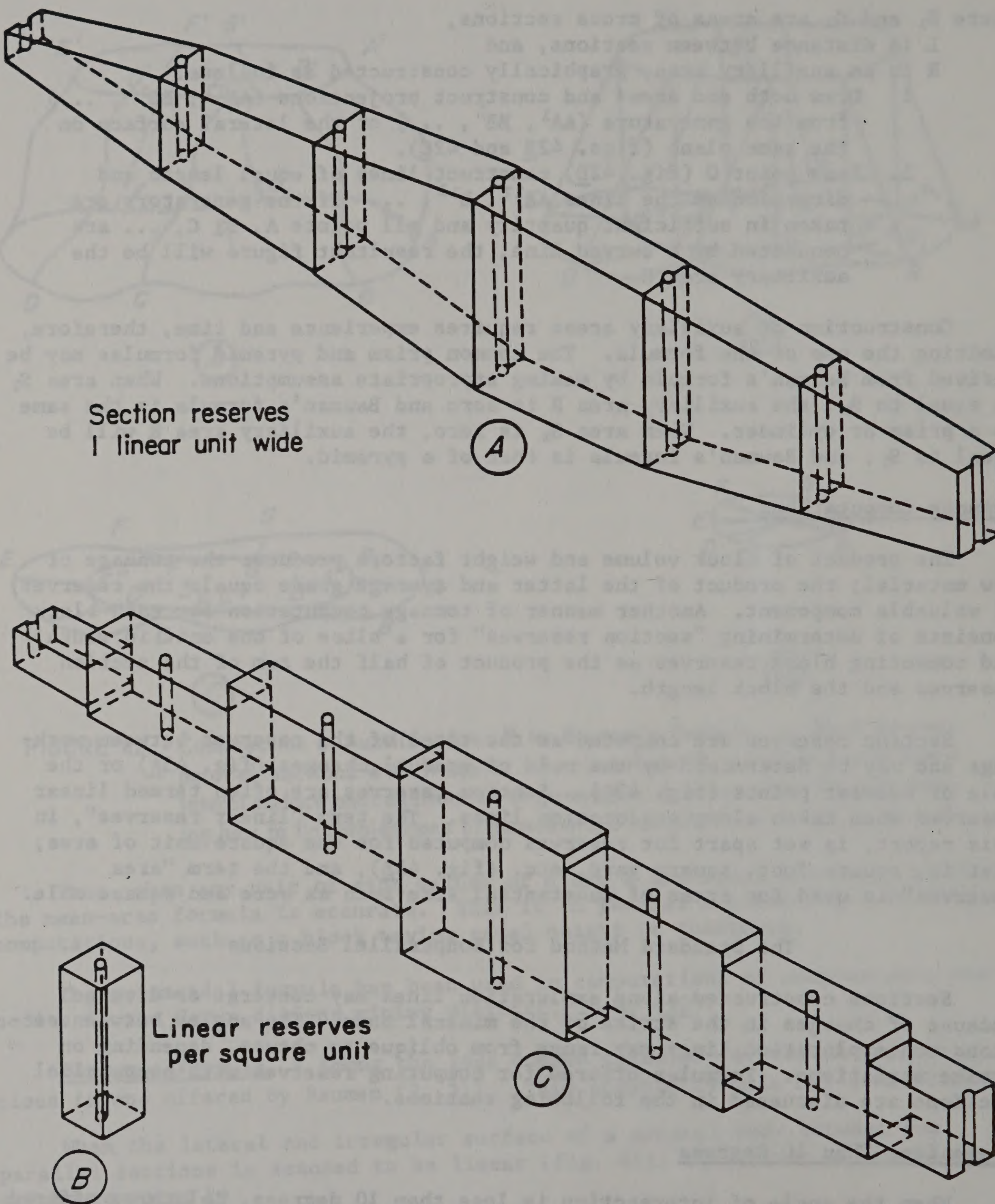


FIGURE 43. - Section and Linear Reserves—Cross-Section Methods. *A*, Layout of auxiliary blocks according to the rule of gradual changes; *B*, linear reserves per square unit; *C*, layout of auxiliary blocks according to the rule of nearest points.

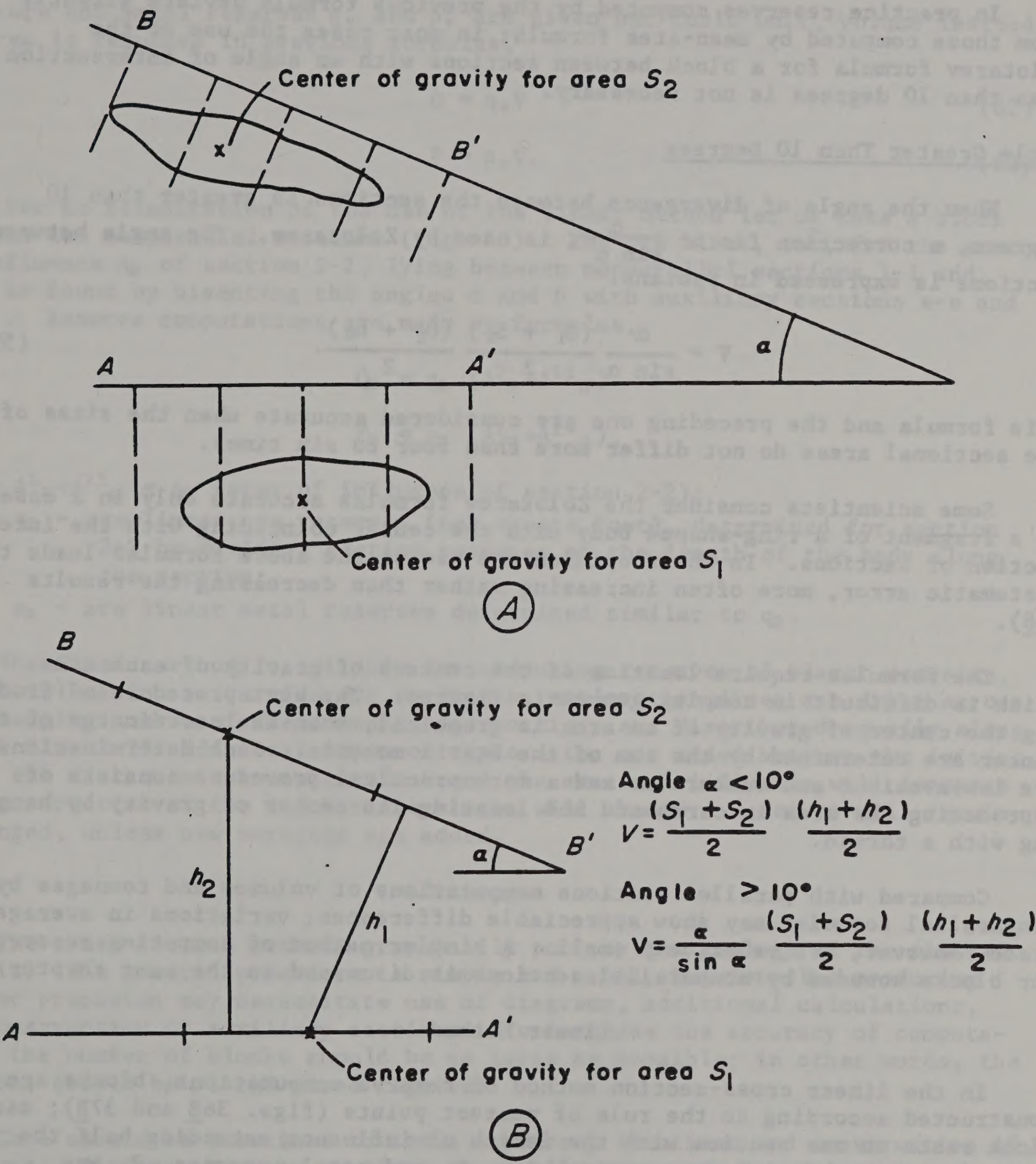


FIGURE 44. - Standard Cross-Section Method for Volume Computations—Nonparallel Sections. *A*, Graphic representation of mineral body crossed by nonparallel sections; *B*, plan—construction of perpendiculars h_1 and h_2 from center of gravity of one section to the other.

where S_1 and S_2 are areas of the mineral body in the sections, and h_1 and h_2 are the lengths of two respective perpendiculars dropped from the center of gravity of one section to another.

In practice reserves computed by the previous formula deviate slightly from those computed by mean-area formula; in most cases the use of the Zolotarev formula for a block between sections with an angle of intersection less than 10 degrees is not necessary.

Angle Greater Than 10 Degrees

When the angle of divergence between the sections is greater than 10 degrees, a correction factor $\frac{\alpha}{\sin \alpha}$, is used by Zolotarev. The angle between sections is expressed in radians.

$$V = \frac{\alpha}{\sin \alpha} \frac{(S_1 + S_2)}{2} \frac{(h_1 + h_2)}{2}. \tag{59}$$

This formula and the preceding one are considered accurate when the sizes of the sectional areas do not differ more than four to six times.

Some scientists consider the Zolotarev formulas accurate only in a case of a fragment of a ring-shaped body with the center coinciding with the intersection of sections. In their opinion the use of the above formulas leads to systematic error, more often increasing rather than decreasing the results (48).

The formulas require location of the centers of gravity of each area, which is difficult in complex geometric figures. The best procedure of finding the center of gravity of an area is graphical, when the coordinates of the center are determined by the sum of the static moments. Such determinations are inconvenient and cumbersome and a more practical procedure consists of reproducing the area in cardboard and locating its center of gravity by hanging with a thread.

Compared with parallel sections computations of volumes and tonnages by nonparallel formulas may show appreciable differences; variations in average grade, however, are relatively small. A simpler method of computing reserves for blocks bounded by nonparallel sections is discussed in the next chapter.

Linear Method

In the linear cross-section method of reserve computations, blocks are constructed according to the rule of nearest points (figs. 36B and 37B); each block rests on one section with the length of influence extending half the distance to the adjoining sections. Ore, Q, and metal reserves, P, are usually determined as the product of linear ore and metal reserves q_L, p_L, and area A,

$$Q = q_L A \tag{60}$$

and

$$P = p_L A. \tag{61}$$

When ore and metal reserves q_v and p_v are given per cubic unit, volume instead of area is required in previous formulas,

$$Q = q_v V \quad (62)$$

and

$$P = p_v V. \quad (63)$$

For an illustration of the use of the linear method let us take a block between two nonparallel sections (fig. 45). The ore block, two, for the area of influence A_2 of section 2-2, lying between nonparallel sections 1-1 and 3-3, is found by bisecting the angles α and β with auxiliary sections e-e and e_1-e_1. Reserve computations are made by formulas,

$$Q_2 = q_2 \ (A^1{}_2 + A^{11}{}_2)$$

and

$$P_2 = p_2 \ (A^1{}_2 + A^{11}{}_2),$$

where $A^1{}_2 + A^{11}{}_2 = A_2$ (area of influence of section 2-2);

q_2 - are linear ore reserves (per square foot), determined for section 2-2 by dividing section reserves on the length of the body along the section;

p_2 - are linear metal reserves determined similar to q_2.

The linear method is suitable for computing reserves of placer deposits, where exploration is carried out in stages; exploration lines are drawn across the changing course of the deposit, and workings are distributed equally along such a line. If additional lines of exploration are added between the initial ones, the distances between sections and the areas of influence will decrease and construction of the appropriate blocks will change. Reserves will remain unchanged, unless new workings are added.

Advantages

The cross-section methods graphically portray the geology of the mineral deposit. The general procedure is simple and rapid, but formulas producing greater precision may necessitate use of diagrams, additional calculations, and construction of auxiliary sections. To increase the accuracy of computations the number of blocks should be as large as possible; in other words, the sections should be placed close together.

Care should be exercised to avoid arbitrary locations and construction of sections. Distance between sections is usually governed, in exploration, by the character of the mineral body and the distribution of mineral values. Selection of sections unjustified by exploration data may influence the size of the areas and, in turn, the computations. Construction should not rely on interpretation made over distances unmerited for the given type of deposit. Most of the disadvantages in the use of this method can be avoided by properly planned exploration.

Computations of two or more ore bodies in the sections are possible.

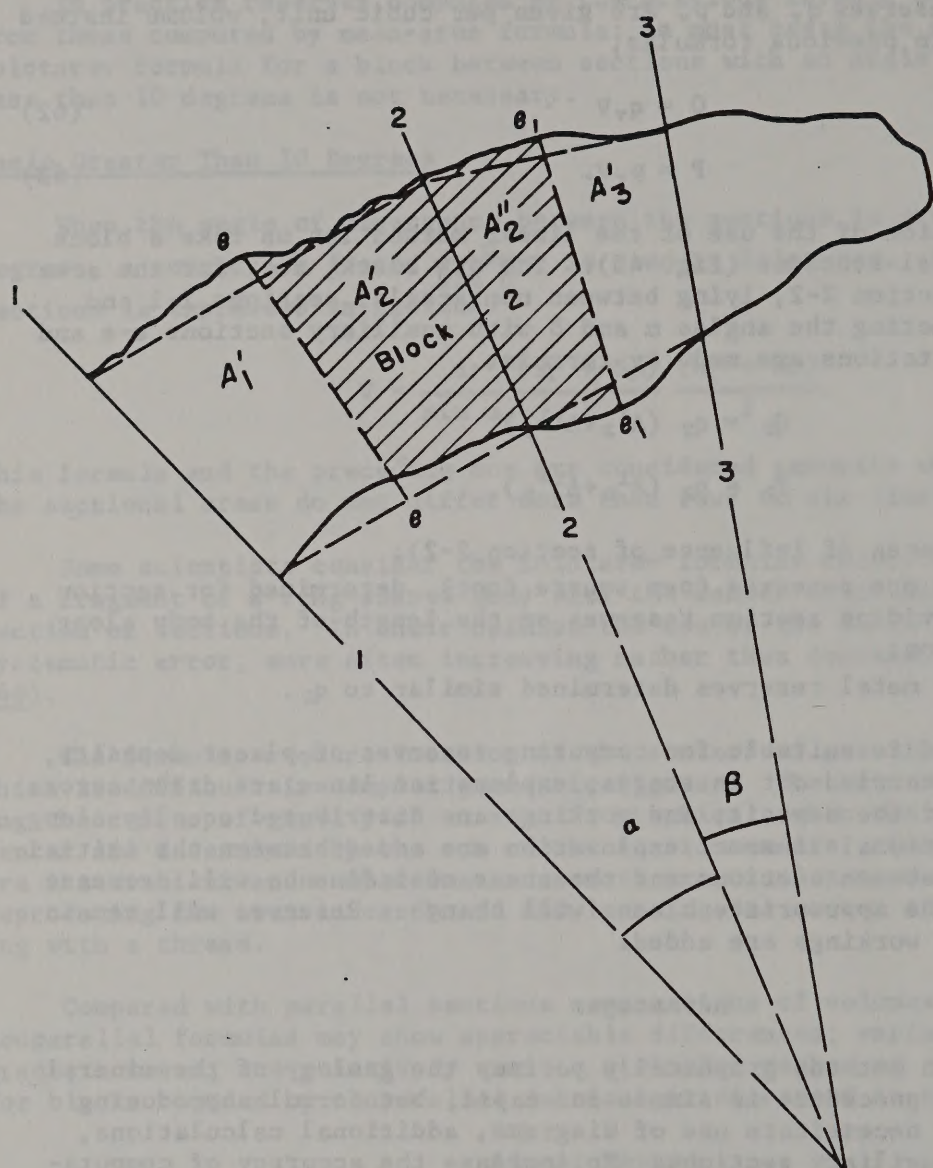


FIGURE 45. - Linear Method—Block Between Nonparallel Sections.

Application

Cross-section methods are being used effectively when construction of sections is possible with a minimum amount of interpolation and extrapolation. The application of various formulas depends on the analysis of the layout of the sections, on the relative size and shape of ore areas, and on the distances between sections.

Common formulas for volume computations are mean-area and frustum. The regular prism formula requires equality in size and shape of both areas. The mean-area formula is accurate when the areas in parallel sections are similar; it should not be used when side dimensions a and b are different. If the areas of parallel sections differ by more than 40 percent, the frustum formula should be used.

The prismoidal formula is accurate for all various forms of solids. Bauman's formula requires construction of an auxiliary area. The proper use of the various formulas for different geometric solids is given in table 12. In general, computation of block volumes by the methods of cross sections requires analysis of the shape and size of sections to determine the best formula, particularly in conjugated blocks.

TABLE 12. - Application of various formulas in computing volumes of solid bodies in standard cross-section method

Name	Formula	Solid bodies betweed parallel sections			
		Prism	Pyramid	Frustum	Wedge
Mean-area............	$L\,\frac{(S_1+S_2)}{2}$	X	-	-	-
Frustum..............	$\frac{L}{3}\,(S_1+S_2+\sqrt{S_1 \times S_2})$	X	X	X	-
Prismoidal (Simpson)..	$\frac{L}{6}\,(S_1+4M+S_2)$	X	X	X	X
Obelisk..............	$\frac{L}{3}\left[S_1 S_2 + \frac{(a_1 b_2 + a_2 b_1)}{2}\right]^1$	X	X	X	X

[1] Reported as a new formula by Kravchenko and Kupfer.

Source: Reference (30).

Well-defined and large bodies that are uniform in thickness and grade or have gradually changing values can generally be computed accurately by cross-section methods. Sections may be vertical, inclined, or horizontal, as in pipelike or stock deposits. Two suites of sections constructed at right angle to each other may be employed for large mineral bodies with more or less evenly distributed values, such as stocks and impregnations. The final results may then be computed as an average of both suites of sections, or one suite may serve for control of the other.

The method should be used with discretion in all cases where the bodies are irregular, or where values tend to concentrate in ore shoots. When computations of several valuable components are required and the mineral body shows grade variations for each component, it is difficult and often impossible to apply cross-section methods. Horizontal cross sections, constructed along levels or horizons of workings, is preferred because of the selection or design of mining method.

Quite often it is necessary to compute reserves of ore between levels, or of different grade within a block, separately. The sum of such auxiliary computations should be equal to the block reserves; absence of such control indicates a possible source of error.

Cross-section methods are easily adaptable for use simultaneously with other methods. Reserves, developed in upper levels by underground workings, may be computed by the mining blocks method, and reserves of lower levels, explored by drilling, by the standard cross-section method. Numerous examples of such computations are described in the literature (60-61, 65).

Method of Isolines

Principles and Formulas

Isolines are curved lines joining all points of equal unit value. They are used to graphically illustrate natural physical and chemical properties or processes that can be expressed by unit values. A common example is a topographic map, where relief is expressed by contours of equal elevation. Isolines are widely accepted in earth and engineering sciences for visual delineation and distribution studies of various physical and chemical phenomena. Well-known applications are maps using lines to depict similar relationships such as isothermal, isostatic, isomagmatic, isopach (or isothickness), isocal (isocalorific values for coal), isocarb (equal content of fixed carbon), isograde, and others. Less common are complex isolines such as linear reserves (foot-percent, tons, per square foot, or dollars per square foot) used in computing reserves of mineral deposits.

The method is based on the assumption that unit values, from one point to another, undergo continuous and uninterrupted changes according to the rule of gradual changes. To construct isolines, intermediate values are determined by interpolation between points of known values; as a result certain properties of mineral bodies may be presented graphically on a plan or section by a system of isolines. In aggregate such a system constitutes an imaginary surface similar to a topographic map. These "toposurfaces" are graphic expressions of numbers and, thus, may be used according to the principles of solid and analytical geometry.

The theory of the method of isolines for use in mining and engineering was developed by Sobolevsky (37, 53). He disclosed that the toposurfaces can be added, subtracted, multiplied, and divided and that even more complex operations, such as extraction of roots, involution, differentiation, and integration, could also be made. Practical applications of this method in geology and mining are varied. Detailed discussion of these applications is beyond the scope of this paper.

Common cases are computations of average thickness, average grade, and average value of a mineral deposit from appropriate isoline maps (6, 11). Only an isopach map is needed to compute volume and tonnage of mineral ore reserves. A unit of volume reserves at a given point on such a map is a product of height, equal to the thickness of the body, and area, equal to a unit value (square foot, square yard, square meter, etc.). A unit of tonnage ore reserves is a division of volume reserves and volume-tonnage factor. To compute the weight of metal or other valuable component in the deposit, isolines of linear metal reserves (product of linear ore reserves and grade) are constructed.

Let us examine a portion of an isopach map (fig. 46). The mineral body, confined in nature by irregular surfaces, is transformed for computations to an equivalent body limited on one side by a flat plane base and on the other by a complex surface represented by a series of isolines of equal thickness or height. Thus, the isopach map gives a distorted picture of the mineral

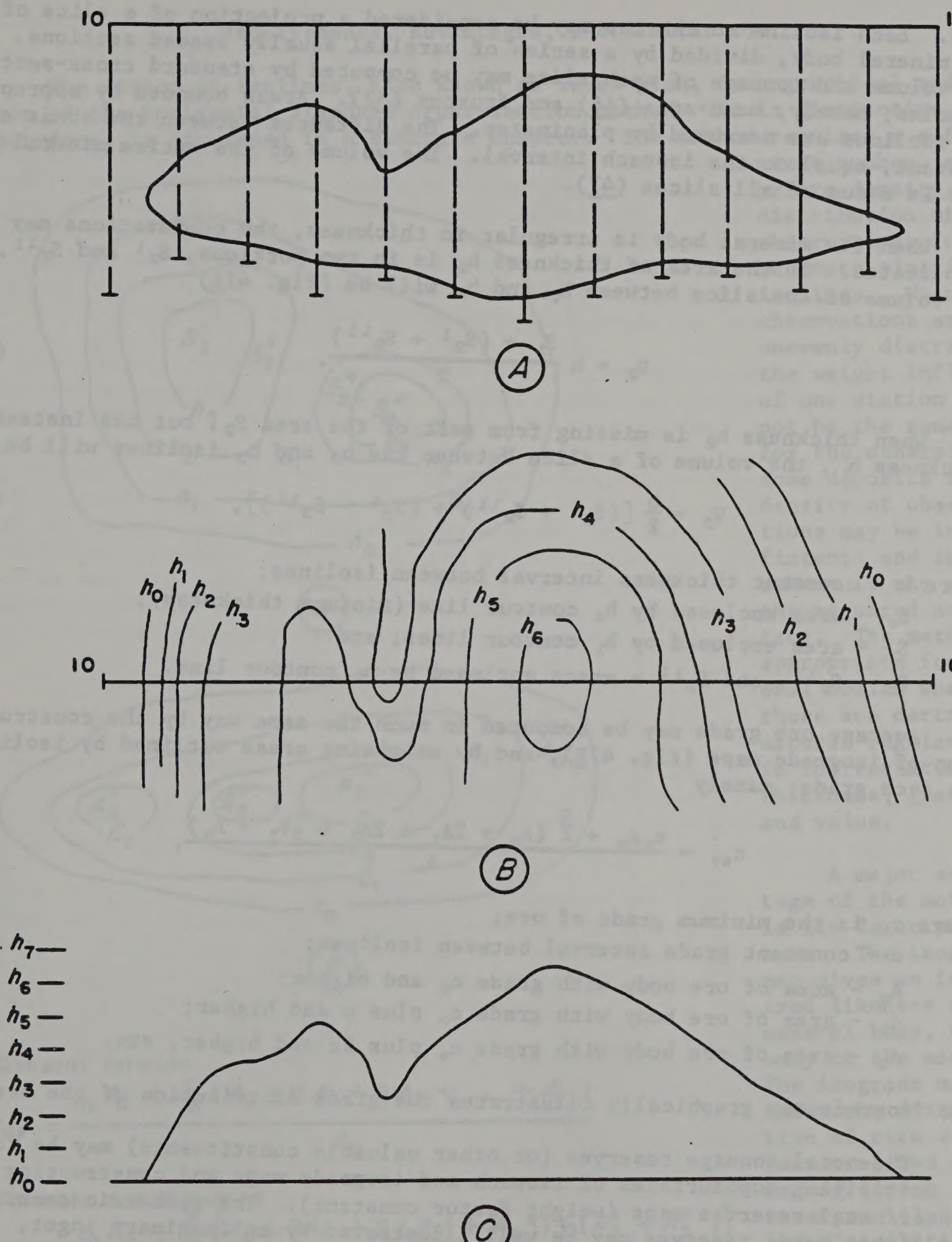


FIGURE 46. - Method of Isolines. *A*, Cross section of a mineral body along exploration line 10-10; *B*, isopach plan of the same mineral body along the exploration line 10-10; *C*, cross section along exploration line 10-10 made from isopach plan *B*.

body. Each isoline on the map may be considered a projection of a slice of the mineral body, divided by a series of parallel equally spaced sections. The volume and tonnage of each slice may be computed by standard cross-section formulas; namely, mean-area (46) and frustum (52). Areas bounded by appropriate isolines are measured by planimeter. The distances between the areas are constant, equal to the isopach interval. The volume of the entire mineral body is a sum of all slices (47).

When the mineral body is irregular in thickness, the computations may be complicated. If the area of thickness h_2 is in two portions, $S_2{}^1$ and $S_2{}^{11}$, the volume of the slice between h_1 and h_2 will be (fig. 47A)

$$V_2 = h \frac{S_1 + (S_2{}^1 + S_2{}^{11})}{2}. \qquad (64)$$

When thickness h_2 is missing from part of the area S_2, but has instead a thickness h_1, the volume of a slice between the h_2 and h_3 isolines will be

$$V_3 = \frac{h}{2} [(S_2{}^1 + S_2{}^{11}) + (S_3{}^1 - S_3{}^{11})], \qquad (65)$$

where h - constant thickness interval between isolines;
S_o - area enclosed by h_o contour line (minimum thickness);
S_1 - area enclosed by h_1 contour lines; and
$S_2{}^1$, $S_2{}^{11}$, and $S_3{}^{11}$ - areas enclosed by h_2 contour line.

Average ore grade may be computed in much the same way by the construction of isograde maps (fig. 47B), and by weighting areas outlined by isolines for each grade; namely

$$c_{av} = \frac{c_o A_o + \frac{c}{2} (A_o + 2A_1 + 2A_2 + \ldots + A_n)}{A_o}, \qquad (66)$$

where c_o is the minimum grade of ore;
c - constant grade interval between isolines;
A_o - area of ore body with grade c_o and higher;
A_1 - area of ore body with grade c_o plus c and higher;
A_2 - area of ore body with grade c_o plus 2c and higher, etc.

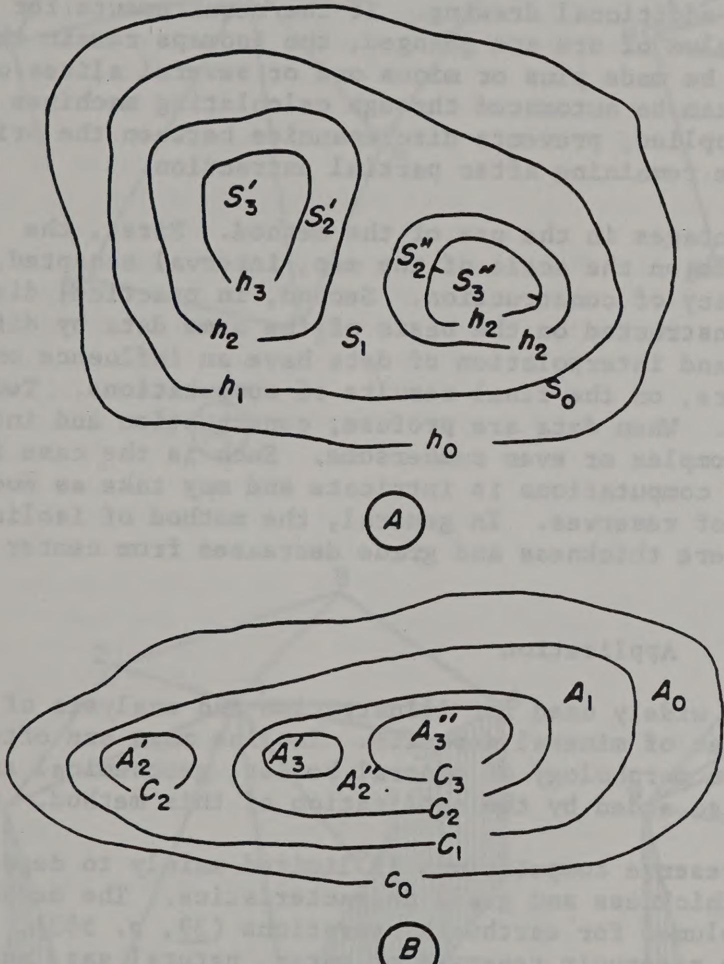
The isograde map graphically illustrates the grade distribution of the ore.

The metal tonnage reserves (or other valuable constituents) may be found by multiplying toposurfaces of isopach and isograde maps and constructing linear metal reserves maps (weight factor constant). The geometric meaning of the linear metal reserves may be well illustrated by an imaginary ingot, received by settling at right angles all metal particles on the plane of the map. The metal reserves of the mineral body then will be confined between the base plane and linear metal toposurface. By slicing the ingot by sections parallel to the base on equal distances, traces of such intersections with toposurface will yield the isolines of linear metal reserves.

Requirements, Advantages, and Limitations

The method of isolines, also known as level plan, is a graphical modification of the horizontal standard cross-section method. It is distinctive and, therefore, is discussed in a separate chapter. The method requires a sufficient number, appropriate density, and distribution of observations for accurate plotting of isolines. When observations are unevenly distributed, the weight influence of one station may not be the same as for the others; in some deposits the density of observations may be insufficient; and in others it may exceed the required accuracy. The method is appropriate for mineral bodies where there are certain natural regularities in the variations in thickness, grade, and value.

A major advantage of the method is its descriptiveness. The isopach map gives an idealized likeness of the mineral body, second only to the model. The isograde map shows the distribution of rich and poor ore, and the map of linear reserves illustrates the distribution of reserves of raw material and valuable constituents. Isoline maps are



General formula

$$c_{av} = \frac{c_0 A_0 + \frac{c}{2}(A_0 + 2A_1 + 2A_2 + \ldots + A_n)}{A_0}$$

Formula for figure *B*

$$c_{av} = \frac{c_0 A_0 + \frac{c}{2}\left[A_0 + 2A_1 + 2(A_2' + A_2'') + (A_3' + A_3'')\right]}{A_0}$$

FIGURE 47. - Isopach and Isograde Maps for Reserve Computations—Method of Isolines. *A*, Isopach map; *B*, isograde map.

easy to read, measure, and interpolate; calculations are replaced by graphic interpretations and there are fewer blocks, instead of numerous small blocks used in some methods.

The method permits better mine planning. The boundaries of cutoff ore are easily constructed and changed. Volume can be computed by measuring areas of respective isolines without additional drawing. If the requirements for minimum grade, thickness, or value of ore are changed, the isomaps remain the same; reserve computations can be made plus or minus one or several slices of the mineral body. The method can be automated through calculating machines or computers and, when properly applied, prevents discrepancies between the original reserve estimates and those remaining after partial extraction.

There are several disadvantages in the use of the method. First, the position of the isolines depends on the scale of the map, interval accepted, density of workings, and accuracy of construction. Second, in practice, dissimilar toposurfaces can be constructed on the basis of the same data by different persons. Construction and interpolation of data have an influence on the size of areas and, therefore, on the final results of computations. Two or more solutions are possible. When data are profuse, construction and interpretation of isolines may be complex or even cumbersome. Such is the case in multimetal deposits. Checking computations is intricate and may take as much time as a complete reestimate of reserves. In general, the method of isolines is best applied to deposits where thickness and grade decreases from center to periphery.

Application

The method of isolines is widely used for illustration and analysis of physical and chemical properties of mineral deposits. Isoline maps are often irreplaceable in studies of the morphology of mineral bodies; geochemical and geophysical prospecting are also aided by the application of this method.

The use of isolines for reserve computations is limited mainly to deposits showing orderly changing thickness and grade characteristics. The method is widely used in computing volumes for earthwork operations (39, p. 583), stockpiles (44), and computing reservoir reserves of water, natural gas, and oil (3, pp. 91-98). It is time consuming and nonoperative when the grid of exploration workings is sparse. It is impractical in structurally broken and small ore bodies, in complex multimetal and very irregular mineral deposits.

Isolines are known to be drawn from interpretation of aerial photographs for computing volume of material removed from the area, as well as ore reserve in stockpiles. Few examples of the use of this method for reserves computations are described in literature (26, 56).

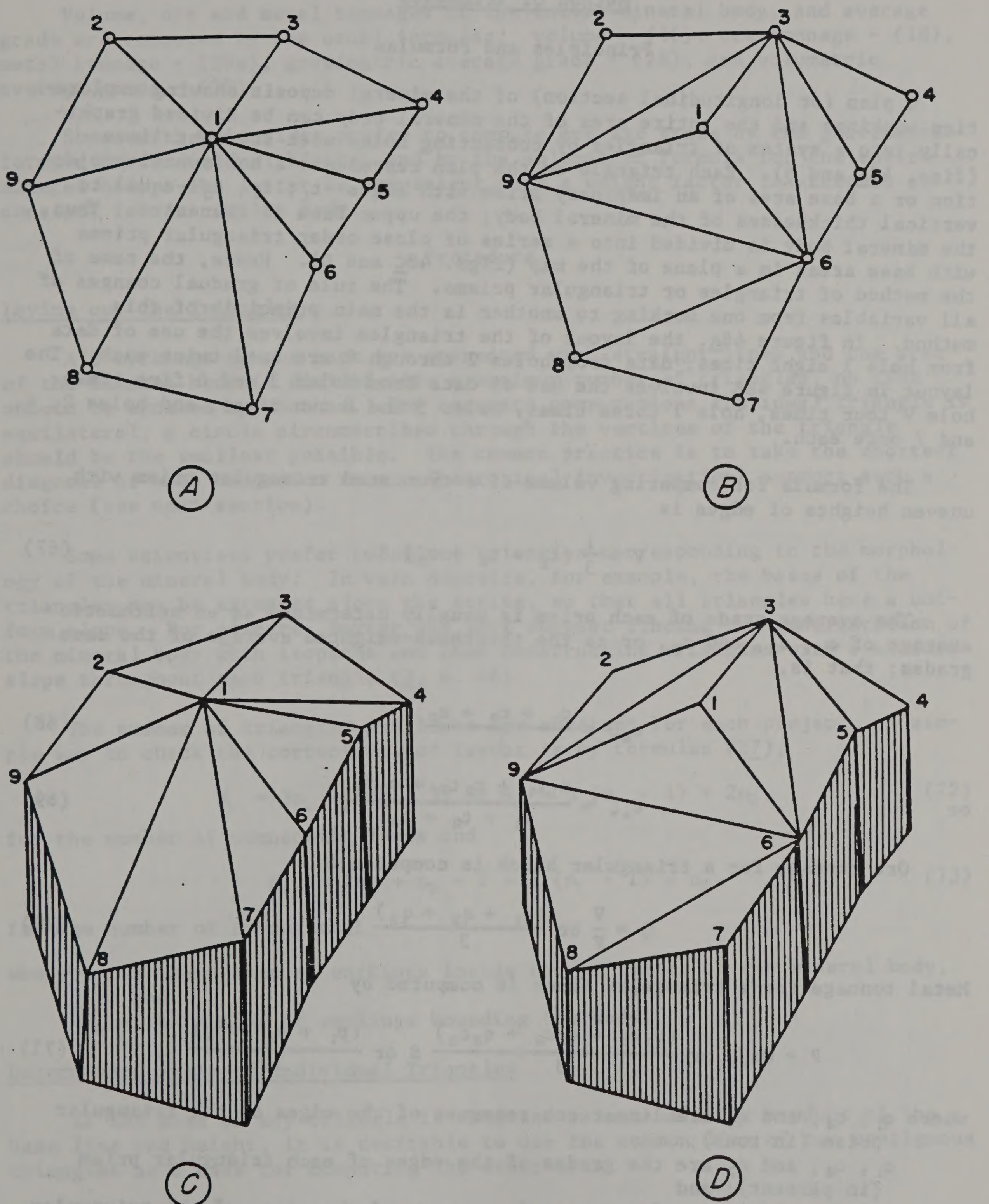


FIGURE 48. - Method of Triangles. *A-B*, Laying out triangles by various methods; *C-D*, Isometric drawing of triangular prisms.

Method of Triangles

Principles and Formulas

A plan (or longitudinal section) of the mineral deposit showing exploration workings and the entire area of the mineral body can be divided graphically into a system of triangles by connecting holes with straight lines (figs. 48A and B). Each triangle on the plan represents a horizontal projection or a base area of an imaginary prism with edges--t_1, t_2, t_3--equal to vertical thicknesses of the mineral body; the upper base is truncated. Thus, the mineral body is divided into a series of close order triangular prisms with base areas in a plane of the map (figs. 48C and D). Hence, the name of the method of triangles or triangular prisms. The rule of gradual changes of all variables from one working to another is the main principle of this method. In figure 48A, the layout of the triangles involves the use of data from hole 1 eight times; data from holes 2 through 9 are used twice each. The layout in figure 48B involves the use of data from holes 3 and 6 five times, hole 9 four times, hole 1 three times, holes 5 and 8 two times, and holes 2, 4, and 7 once each.

The formula for computing volume of a truncated triangular prism with uneven heights of edges is

$$V = \frac{1}{3}(t_1 + t_2 + t_3)\ S. \qquad (67)$$

The average grade of each prism is usually determined as an arithmetic average of c_1, c_2, and c_3, or as the thickness-weighted average of the same grades; that is,

$$c_{av} = \frac{c_1 + c_2 + c_3}{3} \qquad (68)$$

or

$$c_{av} = \frac{c_1 t_1 + c_2 t_2 + c_3 t_3}{t_1 + t_2 + t_3}. \qquad (69)$$

Ore tonnage for a triangular block is computed by

$$Q = \frac{V}{F} \text{ or } \frac{(q_1 + q_2 + q_3)}{3}\ S. \qquad (70)$$

Metal tonnage for a triangular block is computed by

$$P = Qc_{av} \text{ or } \frac{(q_1 c_1 + q_2 c_2 + q_3 c_3)}{3}\ S \text{ or } \frac{(p_1 + p_2 + p_3)}{3}\ S, \qquad (71)$$

where q_1, q_2, and q_3 are linear ore reserves of the edges of the triangular prism (in tons);

c_1, c_2, and c_3 are the grades of the edges of each triangular prism (in percent); and

p_1, p_2, and p_3 are linear metal reserves of the edges of the triangular prism (in tons).

Volume, ore and metal tonnages of the entire mineral body, and average grade are computed by the usual formulas: volume - (16), ore tonnage - (18), metal tonnage - (29a), gravimetric average grade - (28), and volumetric average grade - (27).

Some earth scientists prefer to compute average grade by the gravimetric formula for individual blocks, and by the volumetric formula for the entire mineral body. The latter is convenient when a weight factor is accepted as constant for the entire body.

Procedure

Laying out the Triangles

All workings on the maps are connected with straight lines and the area of the mineral body is divided into a maximum number of triangles; no line should be crossed by another. For accurate computations the ideal triangle is equilateral; a circle circumscribed through the vertices of the triangle should be the smallest possible. The common practice is to take the shortest diagonal of each trapezium area. Theoretical investigations support such a choice (see next section).

Some scientists prefer to select triangles corresponding to the morphology of the mineral body. In vein deposits, for example, the bases of the triangles may be extended along the strike, so that all triangles have a uniform slope. For placers, Baxter and Park suggest finding the configuration of the mineral body with isopachs and then constructing triangles with a uniform slope throughout each triangle (3, p. 46).

The number of triangles and lines are constant for each project. A simple way to check the correctness of layout is by formulas (27),

$$N_L = 3n_1 + 2n_2 + 3, \text{ or } N_L = 3(n_1 - 1) + 2n_2 \tag{72}$$

for the number of connecting lines and

$$N_{tr} = 2n_1 + n_2 - 2 = 2(n_1 - 1) + n_2 \tag{73}$$

for the number of triangles,

where n_1 is the number of workings inside the perimeter of the mineral body, and

n_2 - the number of workings bounding the body.

Determining Areas of Individual Triangles

If the area of any triangle is computed as one-half the product of the base line and height, it is desirable to use the common line of two contiguous triangles as a base for computing the other area.

The procedure of computing volume, ore and metal tonnages, and average grade is illustrated in tables 13 and 14.

TABLE 13. - Determination of average arithmetic grade for each triangle by triangular method

Workings (n), numbers	Triangles numbers	Area (S), sq ft	Thickness (t), ft	Average thickness (t_{av}), ft	Volume (V), cu ft	Weight factor (F), cu ft/ton	Raw material reserves (Q), ton	Grade (c), percent	Valuable component: Average grade (c_{av}), percent	Valuable component: Reserves (P), tons
1........ 2........ 3........	1	S_1	t_1 t_2 t_3	$\frac{t_1+t_2+t_3}{3}$	V_1	F	Q_1	c_1 c_2 c_3	$\frac{c_1+c_2+c_3}{3}$	$P_1 = c^1{}_{av}\ Q_1$
1........ 2........ 4........	11	S_{11}	t_1 t_2 t_4	$\frac{t_1+t_2+t_4}{3}$	V_{11}	F	Q_{11}	c_1 c_2 c_4	$\frac{c_1+c_2+c_4}{3}$	$P_{11} = c^{11}{}_{av}\ Q_{11}$
. ...	-	-	-	-	-	-	-	-	-	-
. ...	-	-	-	-	-	-	-	-	-	-
. ...	-	-	-	-	-	-	-	-	-	-
	N	S_n	-	-	V_n	F	Q_n	-	-	P_n
Total..		$\sum_{i=1}^{n} S$			$\sum_{i=1}^{n} V$		$\sum_{i=1}^{n} Q$			$\sum_{i=1}^{n} P$
Average				$\sum_{i=1}^{n} V / \sum_{i=1}^{n} S$					$\sum_{i=1}^{n} P / \sum_{i=1}^{n} Q$	

TABLE 14. - Determination of thickness-weighted average grade for each triangle by triangular method

Workings (n), numbers	Triangles numbers	Area (S), sq ft	Thick-ness (t), ft	Average thickness (t_{av}), ft	Volume (V), cu ft	Weight factor (F), cu ft/ton	Raw material reserves (Q), ton	Grade (c), percent	Product cxt	Valuable component: Average grade (c_{av}), percent	Valuable component: Reserves (P), tons
1........ 2........ 3........	1	S_1	t_1 t_2 t_3	$\frac{t_1+t_2+t_3}{3}$	V_1	F_1	Q_1	c_1 c_2 c_3	$c_1 t_1$ $c_2 t_2$ $c_3 t_3$	$\frac{c_1 t_1+c_2 t_2+c_3 t_3}{t_1+t_2+t_3}$	$P_1 = c^1{}_{av}\ Q_1$
1........ 2........ 4........	11	S_{11}	t_1 t_2 t_4	$\frac{t_1+t_2+t_4}{3}$	V_{11}	F_{11}	Q_{11}	c_1 c_2 c_3	$c_1 t_1$ $c_2 t_2$ $c_4 t_4$	$\frac{c_1 t_1+c_2 t_2+c_4 t_4}{t_1+t_2+t_4}$	$P_{11} = c^{11}{}_{av}\ Q_{11}$
. ...	-	-	-	-	-	-	-	-	-	-	-
. ...	-	-	-	-	-	-	-	-	-	-	-
. ...	-	-	-	-	-	-	-	-	-	-	-
N........	-	S_n	-	-	V_n	F_n	Q_n	-	-	-	P_n
Total..		$\sum_{i=1}^{n} S$			$\sum_{i=1}^{n} V$		$\sum_{i=1}^{n} Q$				$\sum_{i=1}^{n} P$
Average				$\sum_{i=1}^{n} V / \sum_{i=1}^{n} S$		$\sum_{i=1}^{n} V / \sum_{i=1}^{n} Q$				$\sum_{i=1}^{n} P / \sum_{i=1}^{n} Q$	

Studies by Different Authors

For many years the triangular method was considered standard, although errors in results due to the manner of dividing the area into triangles were recognized. The unreliability of this method was discussed by Harding in 1921 (13-14) and by Zhuravsky in 1934 (68-69). The latter studied the relative error for a volume of a block, explored by four vertical holes, with thicknesses t_1, t_2, t_3, and t_4 and base area S (fig. 49). The volume of the right prism may be computed in two ways: V_1--as the volumes of two triangular prisms with bases ABD and BDC; or V_2--as volumes of two triangular prisms with bases ABC and ADC; that is

$$V_1 = \frac{1}{3}(t_1 + t_2 + t_4)\frac{S}{2} + \frac{1}{3}(t_2 + t_3 + t_4)\frac{S}{2}$$

$$= \frac{1}{6}(t_1 + 2t_2 + t_3 + 2t_4)\,S$$

$$V_2 = \frac{1}{6}(2t_1 + t_2 + 2t_3 + t_4)\,S.$$

In the first case t_2 and t_4, and in the second case t_1 and t_3, are taken twice in each formula. Graphically, the lower surface of the block in the first case is convex and the volume is overestimated, and in the second, the block is concave and the volume is underestimated.

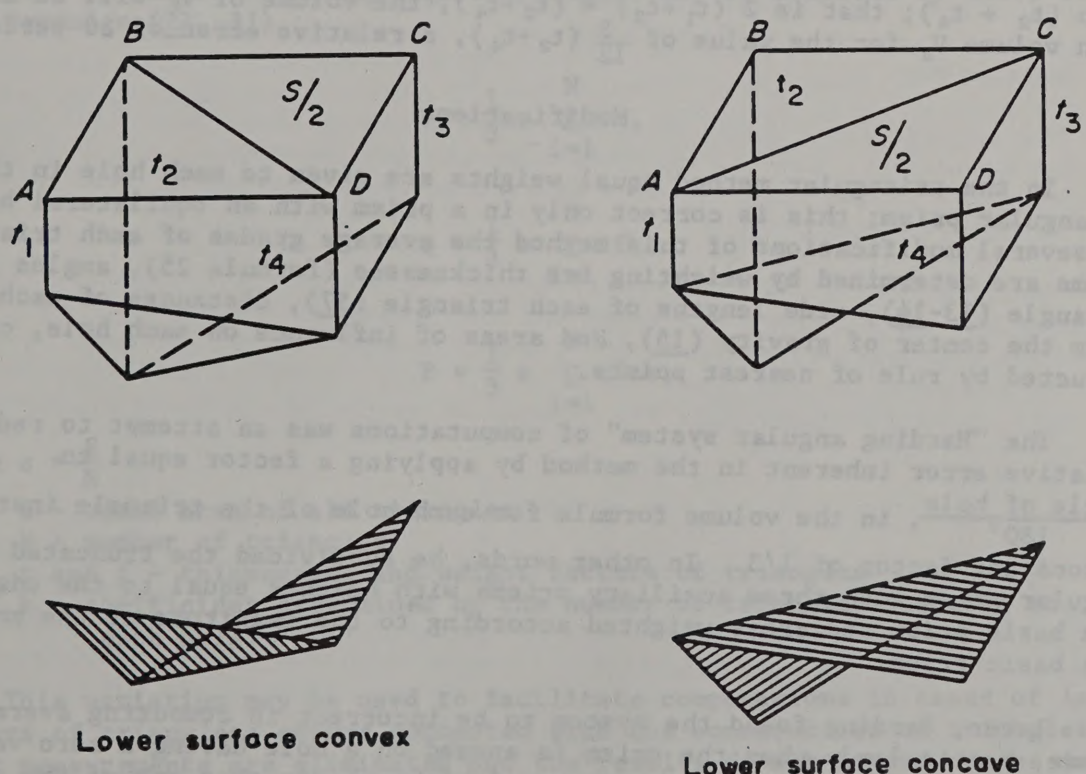


FIGURE 49. - Two Manners of Construction of Triangular Prisms for a Rectangular Prism.

The volume of the prism is computed by half the sum of both cases,

$$V = \frac{V_1 + V_2}{2}, \text{ or } \frac{1}{2}\Big[\frac{1}{6}(t_1 + 2t_2 + t_3 + 2t_4)\, S$$

$$+ \frac{1}{6}(2t_1 + t_2 + 2t_3 + t_4)\, S\Big] = \frac{1}{4}(t_1 + t_2 + t_3 + t_4)\, S.$$

The latter is a standard rectangular prism formula.

The relative error between volumes V_1 and V_2 is

$$\Delta V = V_1 - V_2 = \frac{1}{6}(t_1 + 2t_2 + t_3 + 2t_4)\, S - \frac{1}{6}(2t_1 + t_2 + 2t_3 + t_4)\, S$$

or $$\Delta V = \pm \frac{S}{6}(t_1 - t_2 + t_3 - t_4). \qquad (74)$$

If ΔV is equal to zero, the volumes V_1 and V_2 are equal, and

$$t_1 + t_3 = t_2 + t_4. \qquad (75)$$

In other words, the method of triangles is accurate only if the sum of the two opposite edges, t_1 and t_3, of each rectangular prism are equal to the two remaining edges, t_2 and t_4. Assuming that $(t_1 + t_3)$ is two times less than $(t_2 + t_4)$; that is $2(t_1 + t_3) = (t_2 + t_4)$, the volume of V_1 will be more than volume V_2 for the value of $\frac{S}{12}(t_2 + t_4)$, a relative error of 20 percent.

Modifications

In the triangular method equal weights are given to each hole in the triangular prism; this is correct only in a prism with an equilateral base. In several modifications of this method the average grades of each triangular prims are determined by weighting ore thicknesses (formula 25), angles of the triangle (13-14), side lengths of each triangle (57), distances of each hole from the center of gravity (15), and areas of influence on each hole, constructed by rule of nearest points.

The "Harding angular system" of computations was an attempt to reduce the relative error inherent in the method by applying a factor equal to $\frac{\text{angle of hole}}{180°}$, in the volume formula for each hole of the triangle instead of a constant factor of 1/3. In other words, he subdivided the truncated triangular prism into three auxiliary prisms with heights equal to the edges of the basic prism and areas weighted according to the magnitude of the angle of the basic prism.

Later, Harding found the system to be incorrect in computing average grade, particularly when the prism is apexed on a hole having a zero value thickness. The computed grade may be of higher tenor than any of the other values from the holes of the prism. To correct such errors the following formulas are given by Harding (13, p. 124):

When all holes have positive values,

$$t_{av} = \frac{t_1 + \frac{t_1 + t_2}{2} + \frac{t_1 + t_3}{2}}{3} = \frac{4t_1 + t_2 + t_3}{6}. \quad (76)$$

Similar formulas are used for holes 2 and 3.

When t_3 has zero value,

$$t_{av} = \frac{3t_1 + t_2}{6}; \quad (77)$$

if t_2 and t_3 have zero values,

$$t_{av} = \frac{t_1}{3}. \quad (78)$$

Additional studies of the triangular method led Harding and others to the development of the polygonal method.

When the workings are distributed in a regular grid and the areas of the triangles are equal or nearly equal, reserves may be computed by these simplified formulas (27, 31),

$$V = \frac{1}{3} s \sum_{i=1}^{N} tk, \quad (79)$$

$$Q = \frac{1}{3} s \sum_{i=1}^{N} tfk, \quad (80)$$

and

$$P = \frac{1}{3} s \sum_{i=1}^{N} tfck, \quad (81)$$

where $s = \frac{S}{N}$

S - total area of all triangles
N - number of triangles
t and f - thicknesses and weight factors of triangles
k - coefficient determined by the number of triangles starting at each hole.

This variation may be used to facilitate computations in cases of large numbers of triangles; errors connected with the construction of triangles and their measurements are eliminated and the results do not depend on the manner of constructing the triangles.

Distinctive Features

The method of triangles is basically formal and withdrawn from geologic and mining considerations. The inside perimeter of the net of triangular prisms may be in conflict with the physical boundaries of the body, and the prism sides may cross the boundaries of individual ore types. It is often difficult or even impossible to subdivide the ore body into segments of similar thickness or grade. Triangles may conceal the distribution of variables.

The procedure for reserve computations by the method is relatively simple, when the formula for truncated prisms is used. Modifications of the method require more elaborate computations. The relative error depends on the manner in which the area is divided into triangles, their form, and the total number of triangles.

In comparison with other methods the triangle method requires construction of a greater number of blocks ultimately resulting in labor and time consuming computations. When a mineral body contains several valuable components, computations may also be cumbersome.

In the triangular method of computations the use of exploration data concerning individual mine workings may not be constant, for example, in figure 48<u>A</u> data for holes on the perimeter of the body are used two times in comparison with eight times within the body (hole 1). In cases of irregular sharp changes in variables, inside workings may have a disproportionate influence on the computations.

The method is not exact when variables decrease from the center to the outside boundaries, such as the thickness of lenslike bodies. According to Zhuravsky the volume reserves of a lenticular body computed by this method will be underestimated (<u>68</u>).

Thus, errors in computing reserves may be substantial, particularly when fluctuations of variables are large and the number of triangles is small. When mine workings are numerous and closely spaced, errors for each triangle tend to compensate each other. Even in the most favorable distribution of workings, such as square set, the triangular method may produce an appreciable error.

Application

The uniform and gradual changes of variables are characteristic features of only a few mineral deposits, predominantly sedimentary. Naturally, the triangular method, based on the rule of gradual changes of interpretation of exploration data, is most applicable to such deposits. Large sedimentary and large disseminated ore deposits, explored by regularly spaced drill holes, have been computed by this method (<u>58</u>).

Method of Polygons

Principles

The method of polygons, also known as polygonal prisms, equal spheres of influence, areas of equal influence, and areas of nearest points, is based on the concept that all factors, determined for a certain point of a mineral body, extend half the distance to adjoining and surrounding points, thus forming an area of influence. The rule of nearest points was discussed in part 1 of this report. Briefly, areas of equal influence are found for workings symbolized on the map as points by using perpendicular bisectors and for those symbolized as lines by angle bisectors.

The first description of the method was given as early as 1909 by Boldyrev (57). In the United States the method was developed independently from the triangular method by Davis (13, p. 122) and Harding (14) during the 1920's. A concept of areas of equal influence was introduced step by step, and it was accepted and developed as a new principle for the polygon method, where triangles are used as auxiliary constructions. The first application of this method in the United States was in computing reserves of extremely irregular bodies of the Joplin and Wisconsin zinc deposits in 1920 (13).

Procedure and Construction of Polygons

In this method, the explored portion of the mineral body is substituted by a series of polygonal prisms, the plane bases being equal to areas of influence of appropriate workings (fig. 50). Each such prism assumes the thickness, weight factor, and grade determined for such workings.

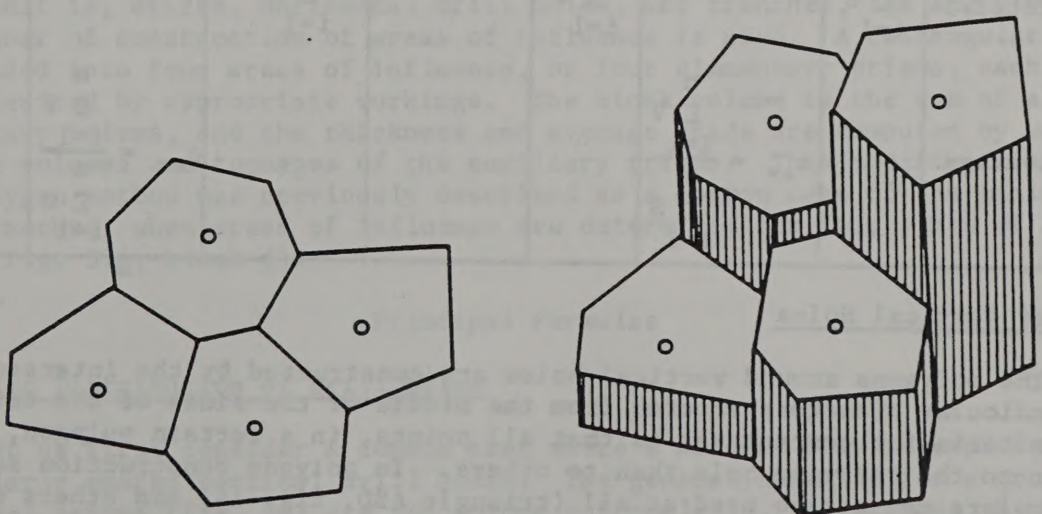


FIGURE 50. - Method of Polygons. Plan and polygonal prisms.

The usual steps in computing reserves by this method are

1. Construction of auxiliary triangles, when necessary; the manner of construction of triangles has no influence on the final shape of polygons;

2. Construction of polygons by following a definite order; for example, clockwise and from periphery to the center of the deposit;

3. Computing reserves for each block (table 15);

4. Grouping of blocks on the basis of evaluation of grade, thickness, linear reserves, reliability, etc., and summarizing and classifying reserves into various categories.

TABLE 15. - Computation of reserves by polygonal method

Polygon number	Area (S), sq ft	Thickness (t), ft	Volume (V), cu ft	Weight factor (F), cu ft/ton	Raw material reserves (Q), tons	Valuable component: Grade (c), percent	Valuable component: Reserves (P), tons
1...........	S_1	t_1	V_1	F	Q_1	c_1	P_1
2...........	S_2	t_2	V_2	F	Q_2	c_2	P_2
3...........	S_3	t_3	V_3	F	Q_3	c_3	P_3
. ...	.	.	.	.	.	.	.
. ...	.	.	.	.	.	.	.
. ...	.	.	.	.	.	.	.
N...........	S_n	t_n	V_n	F	Q_n	c_n	P_n
Total..	$\sum_{i=1}^{n} S$		$\sum_{i=1}^{n} V$		$\sum_{i=1}^{n} Q$		$\sum_{i=1}^{n} P$
Average		$t_{av} = \frac{\sum_{i=1}^{n} V}{\sum_{i=1}^{n} S}$				$c_{av} = \frac{\sum_{i=1}^{n} P}{\sum_{i=1}^{n} Q}$	

Case of Vertical Holes

The polygons around vertical holes are constructed by the intersection of perpendicular bisectors erected from the middle of the sides of the triangles. The criteria for correctness is that all points, in a certain polygon, are nearer to the rallying hole than to others. In polygon construction some perpendiculars may not be used at all (triangle ABD, fig. 14) and others may assist construction by their continuation outside the boundaries of the obtuse triangle (triangle BCD, fig. 14).

When the intersection of perpendiculars for two adjoining triangles forms a tetragon, the diagonal line connecting each pair of perpendiculars becomes a side of a polygon. Point O is equidistant from A, D, and C and point O' is equidistant from A, B, and C. The diagonal is equidistant from A and C. The distinctive feature of a correctly constructed polygon is that each of the inside angles between the sides are always less than 180°.

If the angle bisector manner of construction is used in the above example, the property of nearest points is not fulfilled, because most of the auxiliary triangles are not equilateral (fig. 14). In comparison with the perpendicular bisector manner of construction, angle bisectors produce polygons with twice the number of sides. Polygons may be irregular in shape and have internal angles of more than 180°; therefore, they are more difficult to measure with a planimeter. Further analysis shows that polygons constructed by angle bisectors are only another graphic expression of the method of triangles.

In short, construction of polygons by perpendicular bisectors for vertical workings satisfies the principle of nearest points, is simpler, and always the same; area measurements are more accurate.

It is possible to construct an area of influence for a given point by a circle when workings are too widely spaced to safely assume continuity of the mineral body. Circle radius is chosen as optimum for a certain category of reserves and a given type of deposit. In such cases the block is in the form of a cylinder instead of a polygonal prism.

<u>Case of Linear Workings</u>

When a mineral body is explored by workings represented as lines on a plan, that is, drifts, horizontal drill holes, and trenches, the angle bisector manner of construction of areas of influence is used. A rectangular block is divided into four areas of influence, or four elementary prisms, each one characterized by appropriate workings. The block volume is the sum of all elementary prisms, and the thickness and average grade are computed by weighting the volumes and tonnages of the auxiliary prisms. This modification of the polygon method was previously described as a common case of the mining blocks method, when areas of influence are determined for four sides of a block (fig. 31<u>A</u>, block <u>d</u>).

Principal Formulas

<u>Irregular Distribution of Drill Holes</u>

Let us first consider a common case where a mineral body is explored by irregularly spaced vertical drill holes. The general formulas for a group of polygonal prisms (fig. 50) are for volume - formula 16, for average thickness - similar to formula 37, for ore tonnage - formula 18,

for metal tonnage-

$$P = c_1 q_1 + c_2 q_2 + c_3 q_3 + \dots + c_n q_n , \qquad (82)$$

for average grade - formula 29.

Regularly Spaced Drill Holes

More simple formulas may be derived from principal ones when workings are laid down in a regular grid to form simple polygons, such as squares, rectangles, or hexagons. The common nature of these modifications is that the areas of influence for each hole, except those lying on the boundary of the body, are equal in size.

Square Net of Workings. - The volume of an area, s, explored by four holes with thicknesses t_1, t_2, t_3, and t_4 is

$$V = \frac{(t_1 + t_2 + t_3 + t_4)}{4} s. \qquad (83)$$

The area of mineral bodies that have been explored by numerous holes located in the corners of squares will be divided by perpendicular bisectors into squares with equal areas, s; the formula for volume computation will transform to

$$V = \frac{t_1 + t_2 + t_3 + \ldots + t_n}{N} S = (t_1 + t_2 + t_3 + \ldots + t_n) s, \qquad (84)$$

where t_1, t_2, t_3, ..., t_n are thicknesses of holes, and

$$s = \frac{S}{N}.$$

When the quantity of holes is limited or more accurate results are desirable, the above formula may be improved by adding to each variable a factor based on true areas of influence in arbitrary units, termed "weight." Thus, in a square network, the area of influence of a hole for a complete square must have a weight of four, for a side hole a weight of two, and for a corner hole a weight of one (fig. 51A).

Chessboard Grid. - In a chessboard or triangular grid map, the entire area of the mineral body will be divided by perpendicular bisectors into hexagons with equal areas (fig. 51B). Formulas for computing volume will be the same as a square grid, except--s will equal the area of a hexagon. When the number of workings is limited, the formula should have a weight of six for a complete, three for a half, and one and a half for a corner of a hexagon.

Requirements, Advantages, and Limitations

The method of polygons is based on theoretical assumptions rather than on geologic and mining considerations and requires a suitable plan or longitudinal section. The correct manner of constructing areas of influence requires experience; however, there is only one way to do it, and the results do not depend on personal judgment. In comparison with other methods the nature of the mineral deposit is poorly illustrated, although, polygons may, under appropriate pattern of exploration workings for a given type of deposit, indicate reasonably well the distribution of thick and narrow and high- and low-grade portions of the body.

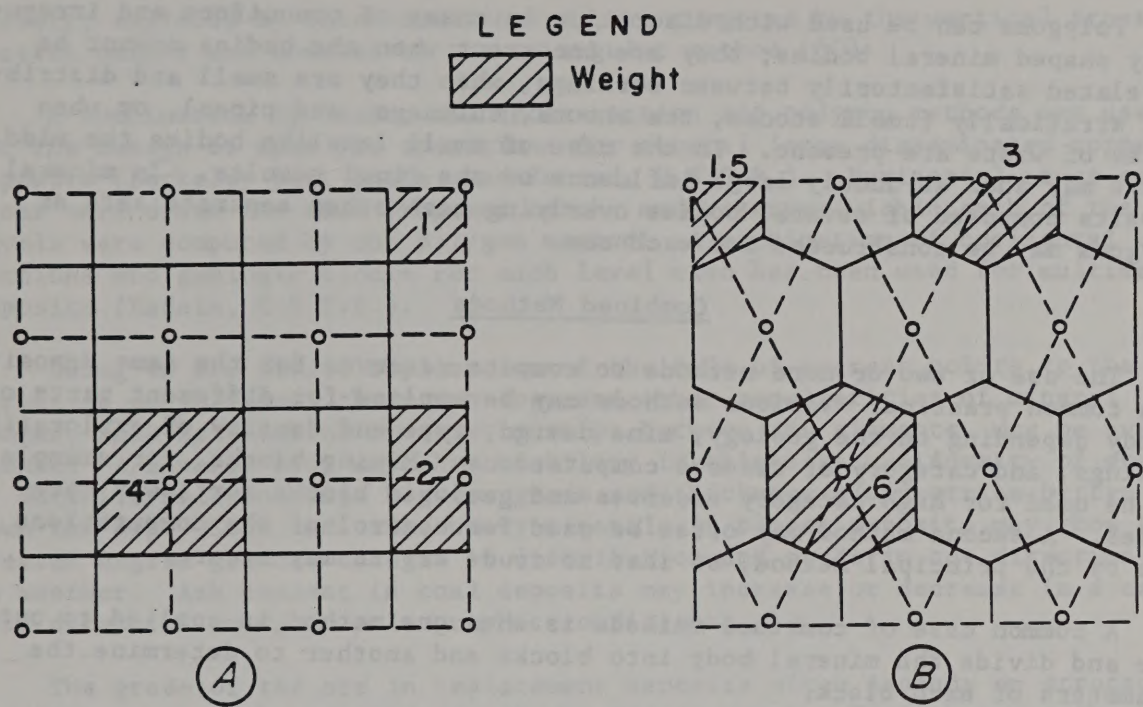


FIGURE 51. - Regularly Spaced Drill Holes—Method of Polygons. *A*, Square net pattern of drilling; *B*, chessboard or triangular pattern of drilling.

The factors, thickness, grade, and weight, are considered constant for each block. Hence, each block is computed without influence from any adjoining blocks, and it is possible to add and compute reserves for new blocks as exploration progresses. In other methods new data often calls for a complete recalculation of reserves.

When the workings are in a regular grid pattern, reserve computations are simplified. The size of the polygons varies when workings are unevenly spaced. More widely spaced holes may have an undue influence on the size of blocks and on the average grade. Any one block may have an unreasonably large influence on the final computations, if the variables of this block vary strongly from the variables of the others.

In the case of irregular distribution of workings it is necessary to measure each polygon with a planimeter. When the blocks are numerous such measurements may be time consuming.

Application

Favorable criteria for the use of the method of polygons are the proven continuity of a mineral body between workings and the gradual changes of all variables. The method is best applied when the workings are numerous and in grid pattern. The greater the number of blocks and the more regular the grid, the more accurate are the computations. Reserves of tabular bodies (beds, blankets, and thick veins), large lenses, and stocks are successfully computed by this method (<u>43</u>, <u>45</u>).

Polygons can be used with discretion in cases of nonuniform and irregularly shaped mineral bodies; they are incorrect when the bodies cannot be correlated satisfactorily between workings, when they are small and distributed erratically (small stocks, ore shoots, chimneys, and pipes), or when horses of waste are present. In the case of small lenslike bodies the middle blocks may show an unduly large influence on the final results. In mineral deposits composed of several bodies overlying each other separate sets of polygons may be constructed for each one.

Combined Methods

The use of two or more methods to compute reserves for the same deposit is a common practice. Various methods may be applied for different parts of a body depending on the geology, mine design, type and density of exploration workings, and category of reserve computations. Mining blocks, for example, may be used for high category reserves and geologic blocks for lower categories. A second method may often be used for control of the computations made by the principal method, so that no crude errors may occur.

A common case of combined methods is when one method is applied to outline and divide the mineral body into blocks and another to determine the parameters of each block.

Methods of mining blocks, geologic blocks, and cross sections have been used for high-grade nonferrous veins developed by underground workings, and the polygon method for disseminated ore bodies (copper-zinc deposits in Butte, and copper-nickel deposits in Noril, U.S.S.R.). The Corrigan-McKinney Steel Company, Michigan and Minnesota, computed the reserves of No. 1 ore body by the average factors and area methods and the No. 2 ore body by the cross-section methods (28, p. 147).

At the Inspiration mine, Arizona, disseminated copper ore reserves based on drilling were computed by polygons (quadrilateral prisms) and checked by the cross-section methods (60). At the Ray Copper mine, Arizona, large reserves of irregular bodies of disseminated ore, were computed by the mining blocks method for the portion of the ore body developed for underground mining and by vertical cross sections for the portion explored by churn and diamond drilling (61).

At the Kennecott Copper mine, Bingham Canyon, Utah, the method of horizontal cross sections was used to separate each proposed bench of the open pit development of a large low-grade disseminated copper body. Mining blocks were used for computing reserves developed by underground workings and polygon method for areas explored by vertical churn drill holes. The triangular prism method was used for computing reserves explored by churn drills below underground workings (58).

At the Copper Queen mine, Bisbee, Ariz., "ore in sight" reserves of an irregular and lenticular limestone-replacement body were computed by horizontal cross sections and probable reserves by two sets of vertical cross sections constructed at right angles to each other. A low-grade copper-porphyry

deposit, explored by churn drill holes, was computed by the vertical cross-section method and checked by the triangular method (50).

A combination of horizontal cross-section and polygon methods was used for the design of open pit operations for several large disseminated copper deposits (Berkeley pit, Butte, and Konrad, U.S.S.R.). Horizontal cross sections were drawn for each proposed bench, and reserves within each of the two levels were computed by the polygon method. A combination of horizontal cross sections and geologic blocks for each level also has been used for multimetal deposits (Kadain, U.S.S.R.).

Owing to the strict application of the rule of nearest points to the construction of polygons, the morphology and other peculiarities of mineral bodies, such as patterns or relationships between the elements, may be overlooked. A mineral body may show stability in thickness, uniformity of grade, or definite relationships between grade and thickness along strike better than down the dip. Gold and other heavy minerals in placer deposits may show to a greater degree gradual changes in distribution and grade in one direction than in another. Ash content in coal deposits may increase or decrease in a certain direction owing to paleographic conditions.

The grade of the ore in replacement deposits often depends on structural and lithological characteristics of the country rock. In such cases the method of polygons may be modified; holes may be first connected with auxiliary lines on the basis of geomorphology and other peculiarities, such as strike, dip, or rake of the ore body. Square and rectangular blocks may then be formed by construction of lines parallel to and/or perpendicular to auxiliary lines. This modification is often described in literature as a method of rectangles, or various areas of influence in contrast with the polygon or equal areas of influence method.

Examples of reserve computations published as rectangular methods are classified in this report according to the accepted principles of interpretation of exploration data; as mining blocks method, in underground mining; and as a simple modification of the polygon method, in a regularly spaced grid of drill holes.

In irregularly spaced drill hole grids, the rectangular block construction is found to be subjective, or affected by personal opinion. The use of such construction shows the same disadvantages and the same limitations as the method of triangles.

Thus, the rectangular system may be considered as a combined method, if block construction is made on the basis of geologic, mining, and economic considerations, rather than on plain geometrical points of view, and the factors are computed by arithmetic average, by thickness-weighted average, or by area-weighted average, determined according to the rule of nearest points.

Source of Errors in Reserve Computations

Combined average error in any reserve computations may be expressed by

$$M_{av} = \sqrt{m_{te}^2 + m_m^2 + m_g^2}, \tag{85}$$

where M_{av} - average relative error of reserve computations;

m_{te} - average relative error of technical errors;

m_m - average relative error of the method and formula used;

m_g - average relative error due to the interpretation of exploration data.

Random errors due to precision of observations compensate each other; they are small in comparison with errors of interpretation and may be disregarded. Biased errors, due to inexperienced personnel, equipment defects, and improper techniques of observations and analyses, may be appreciable and should be compensated for by correction factors prior to computations (part 1).

The comparative accuracy of various methods of computing reserves has been discussed in many publications (15, 46, 49, 53). According to foreign sources the variance in reserves of the same high category computed by different methods for a deposit rarely exceed 10 percent (appendix C).

When the results of computations by a selected method are within 1 to 5 percent of results received by other methods, they are considered by Stammberger to be accurate (59).

In general, the average relative error of the method and formula used for reserve computations should lie within the same limits and should not exceed the average relative error for determining grade, as well as other factors; otherwise such errors will distort the precision of the results of exploration.

In most cases these relative errors due to method are neglected, because the errors of interpretation are much larger and determine the accuracy of computations. The latter errors depend on the type and form of the deposit, on degree of variation in thickness, grade, and other factors, on the kind of exploration workings, their density, and on sample technique.

SUMMARY

Reserve computations of a mineral deposit are a geologic and engineering problem; it is often an intricate task. Selection of a method depends on the geology of the mineral deposit, the kind and density of workings, the appraisal of geologic and exploration data, and the accuracy required. Time and cost of computations are often important considerations.

Knowledge of the mineral deposit's geology is a prerequisite to any reliable computation. This knowledge includes space location, size, shape, environment, country rock, overburden, and hydrology; average grade and distribution of valuable and detrimental constituents; and mineral, chemical and physical characteristics of the raw material.

Accurate computations of a certain deposit require a properly selected and executed exploration program; that is kind of workings, drilling or underground system, number and density of observations, sampling procedure (location, sample interval, and weight of samples), and accurate measurements, analyses, and tests.

To select the best method careful analysis of geology and exploration should be made. In general, the method (or combination of methods) selected should suit the purpose of computations and the required accuracy; it should also best reflect the character of the mineral deposit and the performed exploration. In a complex or irregular deposit, it is advisable to use two or more methods for better accuracy and self-confidence. An average of these methods may be accepted as a final result, or the values of one method may be considered as a control of others.

The purpose of reserve computations is one of the most important consideration in selecting a method. For preliminary exploration the method should best illustrate the deposit, the operations, and permit sequential computations and appraisal. On the other hand, time-consuming procedures should be avoided if reserves are being computed for prospective planning.

The system of mining, or the problem of selecting one, may influence the preference. A certain method of computation may facilitate more so than others the design of development and extraction operations owing to technical and economic factors (mining by levels, average grade, different cutoff, etc.). This explains why, in practice, methods of mining blocks and cross sections are preferred.

The principles of interpretation of exploration data and the analytical perfection of formulas are also considered in method selection. The principles that essentially uphold the described conventional methods are:

Method:	Principles
Analogous	Geologic inference.
Geological blocks	Do.
Mining blocks	Mining and other considerations.
Standard cross section and isolines	Rule of gradual changes.
Triangular prisms	Do.
Linear cross section	Rule of nearest points.
Polygonal prisms	Do.

All formulas for computing volumes, tonnage, and average factors are approximate, because of the irregular size and shape of the mineral body, errors in substituting natural bodies by more simple geometric ones, geologic interpretation, assumptions, and inconsistency in the variables. Accuracy of the final results usually depends more on geologic interpretation and assumptions rather than on the method used. Systematic exploration and uniform

sampling generally simplify the selection and the use of the conventional methods and produce greater accuracy. Reserves of the same category computed by different methods and based on the same data, usually differ slightly.

The average factors and area methods are widely used by earth scientists. In the analogous method, reserves of a block of a deposit can be computed with reasonable accuracy by analysis of results of exploration, development, and extraction (past production) from adjoining blocks of the same or even geologically similar deposits. In this method an individual block of the same or geologically similar deposit may be computed on the basis of limited, or even a single observation properly taken. In the geologic block method, the mineral body is subdivided into segments and blocks essentially on the basis of geology; average factors for each segment or block are determined according to available data by various methods.

The mining blocks method requires adequate data to allow subdivision of the mineral body into blocks either proved or semiproved for extraction. Most often it is used in mining thin and medium-thick veins and tabular bodies.

The cross-section methods are the most convenient ways for computing reserves of uniform mineral deposits. In the standard cross-section method the mean-area formula of a prism with base areas in parallel sections is the most common one; it is precise when there is no large difference in size and shape of base areas. In case of disparity between base areas of more than 40 percent, frustum or prismoidal formulas are used.

In underground mining, horizontal cross sections constructed along the proposed mining levels are often preferred in mine design. Two sets of vertical sections at right angles to each other would better illustrate stocklike bodies than any other method. Computations may be made with the final results taken as half the sum of both suites.

The linear cross-section method, where reserves are first determined for a unit of area, unit of volume, or for the sections, is used with advantage in bedded and placer deposits.

The method of isolines requires numerous observations with data more or less regularly distributed in the vertical or horizontal plane of the mineral deposits. It is applicable to deposits of gradual physical and chemical changes, such as sedimentary deposits. Large placer gold deposits, explored by hundreds of shallow pits or drill holes, may be well illustrated and evaluated by the method of isolines.

The analytical methods (triangular and polygonal prisms) are deficient in exposing the morphology of the mineral body and the fluctuations of variables within the individual blocks. Although average thickness and grade are computed, the pattern of their space distribution is not revealed.

The method of triangles is applicable to a few predominantly sedimentary deposits, the mineralization of which is consistent with the rule of gradual

changes. The method must be carefully applied. Errors of computations may be very high owing to irregularities in variables and unsystematic exploration.

The polygonal method is successfully used in computing reserves of tabular deposits, such as sedimentary beds of coal, phosphate rock, and oil shales; blanket-type, large lenses; and thick vein bodies. The accuracy of the results increases with the number of blocks and the density of the grid of workings and drill holes.

BIBLIOGRAPHY[3] [4]

1. Barishev, N. V. Tochnost' Analiza Prob, Ispol'zuemykh Dlya Podscheta Zapasov (Accuracy of Sample Analyses Used in Computing Reserves). Materialy po Metodam Razvedki i Podscheta Zapasov (Materials for Methods of Exploration and Reserve Estimates). Issue I. Vsesoyuznaya Komissiya po Zapasam Poleznykh Iskopaemykh, Ministerstvo Geologii S.S.S.R. (All-Union Committee on Mineral Reserve, Ministry of Geology, U.S.S.R.), 1948, 43 pp.

2. Bauman, V. I. K Voprosu ob Opredelenii Zapasov Mestorozhdeniy (About the Problem of Computing Mineral Reserves). Gorn. Zhur. (Mining Journal), v. 4, December, 1908, pp. 209-224.

3. Baxter, C. H., and R. D. Parks. Mine Examination and Valuation. Michigan College of Mining and Technology, Houghton, Mich., 2d ed. 1939, 331 pp.

4. Becker, R. M., and Scott W. Hazen, Jr. Particle Statistics of Infinite Populations as Applied to Mine Sampling. BuMines Rept. of Inv. 5669, 1961, 79 pp.

5. Berkenkotter, R. D., and Scott W. Hazen, Jr. Statistical Analysis of Diamond-Drill Sample Data From the Cebolla Creek Titaniferous Iron Deposit, Gunnison County, Colo. BuMines Rept. of Inv. 6234, 1963, 58 pp.

6. Bishop, Margaret S. Subsurface Mapping. John Wiley & Sons, Inc., New York, 1960, 198 pp.

7. Blondel, F., and S. G. Lasky. Mineral Reserves and Mineral Resources. Econ. Geol., v. 51, No. 7, 1956, pp. 686-697.

8. Bush, A. L., and H. K. Stager. Accuracy of Ore-Reserve Estimates for Uranium-Vanadium Deposits on the Colorado Plateau. U.S. Geol. Survey Bull. 1030-D, 1956, pp. 131-148.

9. Feygin, YA. M. Grafical Metod Opredeleniya Ob"yemov Blokov Mezhdu Parallel'nymi Secheniyami (Graphical Method of Determining Block Volumes Between Parallel Sections). Razvedka i Okhrana Nedr (Exploration and Conservation of Mineral Resources), No. 4, 1956, pp. 32-34.

10. Foreman, C. H. Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho. BuMines Inf. Circ. 6232, 1930, 21 pp.

---

[3] Titles enclosed in parenthesis are translations from the language in which the item was published.

[4] Transliteration of the Russian is according to the system given in the GPO Style Manual.

11. Forrester, J. D. Principles of Field and Mining Geology. John Wiley & Sons, Inc., New York, 1946, 647 pp.

12. Freeze, A. C. Use of Data Processing Machines for Calculating Ore Reserves at the Sullivan Mine. Min. Eng., v. 13, No. 4, April 1961, pp. 382-389.

13. Harding, J. E. Calculation of Ore Tonnage and Grade From Drill-Hole Samples. Trans. AIME, v. 66, 1920-21, pp. 117-126.

14. _____. How To Calculate Tonnage and Grade of an Orebody. Eng. and Min. J. Press, v. 116, No. 11, September 1923, pp. 445-448.

15. Hazen, Scott W., Jr. A Comparative Study of Statistical Analysis and Other Methods of Computing Ore Reserves, Utilizing Data From Maggie Canyon Manganese Deposit, Artillery Mountains Region, Mohave County, Ariz. BuMines Rept. of Inv. 5375, 1958, 188 pp.

16. _____. Statistical Analysis of Sample Data for Estimating Ore. BuMines Rept. of Inv. 5835, 1961, 27 pp.

17. _____. Statistical Analysis of Some Sample and Assay Data From Bedded Deposits of the Phosphoria Formation in Idaho. BuMines Rept. of Inv. 6401, 1964, 29 pp.

18. Hazen, Scott W., Jr., and R. D. Berkenkotter. An Experimental Mine-Sampling Project Designed for Statistical Analysis. BuMines Rept. of Inv. 6019, 1962, 111 pp.

19. Hazen, Scott W., Jr., and G. W. Gladfelter. Using Unequal Sample Interval Lengths and Weighted Averages in Estimating Grade of Ore for Bedded Deposits. BuMines Rept. of Inv. 6406, 1964, 23 pp.

20. Herzig, C. S. Mine Sampling and Valuing. Mining and Scientific Press, San Francisco, Calif., 1914, 163 pp.

21. Hewlett, R. F. A Basic Computer Program for Computing Grade and Tonnage of Ore Using Statistical and Polygonal Methods. BuMines Rept. of Inv. 6292, 1963, 20 pp.

22. _____. Application of Simulation in Evaluating Low-Grade Mineral Deposits. BuMines Rept. of Inv. 6501, 1964, 62 pp.

23. _____. Calculating Ore Reserves Using a Digital Computer. Min. Eng., v. 13, No. 1, January 1961, pp. 37-42.

24. _____. Computing Ore Reserves by the Polygonal Method Using a Medium-Size Digital Computer. BuMines Rept. of Inv. 5952, 1962, 31 pp.

25. _____. Computing Ore Reserves by the Triangular Method Using a Medium-Size Digital Computer. BuMines Rept. of Inv. 6176, 1963, 30 pp.

26. Hughes, R. L., Jr. Volume Estimates From Contours. Econ. Geol., v. 54, No. 5, June-July, 1959, pp. 730-737.

27. Izakson, S. S. Metodika Podscheta Zapasov Ugol'nykh Mestorozhdeniy (Methods of Computing Reserves for Coal Deposits). Gosgortekhizdat (State Scientific and Technical Publishers for Mining Literature), Moscow, U.S.S.R., 1960, 370 pp.

28. Jackson, C. F., and J. B. Knaebel. Sampling and Estimation of Ore Deposits. BuMines Bull. 356, 1932, pp. 114-155.

29. Joralemon, I. B. Sampling and Estimating Disseminated Copper Deposits. Trans. AIME, v. 72, 1925, pp. 607-620.

30. Kravchenko, A. Ya., and S. M. Kupfer. Ratsional'nyye Formuly Dlya Opredeleniya Ob"yemov Blokov Pri Podschete Zapasov Metodom Razrezov (Rational Formulas for Computing Block Volumes in Reserve Estimates by Method of Cross Sections). Razvedka i Okhrana Nedr (Exploration and Conservation of Mineral Resources), No. 8, August 1962, pp. 15-19.

31. Kreiter, V. M. Poiski i Razvedka Mestorozhdenii Poleznykh Iskopaemykh (Prospecting and Exploration of Mineral Deposits). Gosudarsvennoe Nauchno-Tekhnicheskoye Izdatel'stvo Literatury po Geologii i Okhrane Nedr (State Scientific and Technical Publishers of Literature in Geology and in Conservation of Mineral Resources), Moscow, U.S.S.R., v. 1, 2d ed., 1960, 322 pp.; v. 2, 2d ed., 1961, 389 pp.

32. Koch, George S., and Richard F. Link. Accuracy in Estimating Metal Content and Tonnage of an Ore Body From Diamond-Drill Hole Data. BuMines Rept. of Inv. 6380, 1964, 24 pp.

33. Lahee, F. H. Field Geology. McGraw-Hill Co., Inc., New York, 4th ed., 1941, 853 pp.

34. McLaughlin, D. H. Geologic Factors in the Valuation of Mines. Econ. Geol., v. 34, No. 6, 1939, pp. 589-621.

35. McKinstry, H. E., Stanley A. Tyler, E. N. Pennebaker, and Kenyon E. Richard. Mining Geology. Prentice-Hall, Inc., New York, 1948, 680 pp.

36. Mead, W. J. The Relation of Density, Porosity, and Moisture to the Specific Volume of Ores. Econ. Geol., v. 3, No. 4, 1908, pp. 319-325.

37. Ogloblin, D. N. Podschet Zapasov Metalla v Rossipi po Metodu Izolinii prof. P. K. Sobolevskogo (Reserve Estimates of Metals in Placers by Method of Isolines, proposed by Prof. P. K. Sobolevsky). Tswetnyye Metally (Nonferrous Metals), No. 7, 1931, pp. 926-949.

38. Olk, J. F. An Open Pit Application of a Digital Computer. Min. Cong. J., v. 48, No. 1, January 1962, pp. 35-37.

39. O'Rourke, C. E. General Engineering Handbook. McGraw Book Co., New York, 2d ed., 1940, 1120 pp.

40. Parsons, A. B. The Porphyry Coppers. American Institute of Mining and Metallurgical Engineers, New York, 1933, 581 pp.

41. Patterson, J. A. Estimating Ore Reserves Follows Logical Steps. Eng. and Min. J., v. 160, No. 9, September 1959, pp. 111-115.

42. Peele, R. Mining Engineer's Handbook. V. 1 and 2, John Wiley & Sons, Inc., New York, 3d ed., 1947.

43. Pod"yakonov, S. A. Metody Podscheta Plastovykh i Rossipnykh Mestorozhdenii (Methods of Computing Gold Reserves for Bedded and Placer Deposits). Gorn. Zhur. (Mining Journal), No. 10, 1928, pp. 729-791.

44. Pohlman, W. E., and H. Jacoby. New Stockpile Measuring Method. Pit and Quarry, April 1959, pp. 88-91.

45. Poston, R. H. Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District. BuMines Inf. Circ. 6160, 1929, 22 pp.

46. Prokof'yev, A. P. Prakticheskiye Metody Podscheta Zapasov Rudnykh Mestorozhdeniy (Practical Methods for Computing Reserves of Ore Deposits). Gosgeolizdat (State Publishers for Geologic Literature), Moscow, U.S.S.R., 1953, 135 pp.

47. _____. Sravnenie Dannykh Geologorazvedochnykh Rabot s Resul'tatami Ekspluatatsii Mestorozhdeniy (A Comparison of Geologic and Exploration Data With the Results of Exploitation Mineral Deposits). Razvedka i Okhrana Nedr (Exploration and Conservation of Mineral Resources), No. 11, 1958, pp. 25-28.

48. Prokof'yev, A. P., and V. V. Stefanovich. O Podschete Zapasov v Blokakh Mezhdu Neparallel'nymi Secheniyami Sposobom A. S. Zolotareva (Reserve Computations for Blocks Between Non-Parallel Sections by Zolotarev Method). Razvedka i Okhrana Nedr (Exploration and Conservation of Mineral Resources), No. 2, 1960, pp. 10-14.

49. Prokop'yev, E. P. Sravnitel'naya Otsenka Raznykh Metodov Podscheta Zapasov Zolota v Rossipnykh Mestorozhdeniyakh (A Comparative Appraisal of Various Methods of Gold Reserve Estimates in Placer Deposits). Gorn. Zhur. (Mining Journal), No. 10, 1925, pp. 837-845.

50. Prouty, R. W., and R. T. Green. Methods of Sampling and Estimating Ore in Underground and Steam-Shovel Mines of Copper Queen Branch, Phelps Dodge Corp. Trans. AIME, v. 72, 1925, pp. 628-639.

51. Rickard, T. A. The Sampling and Estimation of Ore in a Mine. The Engineering and Mining Journal, New York, 1905, 229 pp.

52. Ryzhov, P. A. Matematicheskaya Otsenka Tochnosti Podscheta Zapasov Metodami Sredne-Arifmeticheskim, Mnogougol'nikov i Treugol'nikov (Mathematical Evaluation of the Accuracy of Reserve Estimates by Average Arithmetic, Polygon and Triangle Methods). Tsvetnyye Metally (Nonferrous Metals), Nos. 4-5, 1940, pp. 26-35.

53. _____. Geometriya Nedr (Subsurface Geometry). Izdatel'stvo Nedra (Mineral Resources Publishing Co.), Moscow, U.S.S.R., 1964, 500 pp.

54. Scarborough, J. B. Numerical Mathematical Analysis. The Johns Hopkins Press, Baltimore, Md., 5th ed., 1962, 594 pp.

55. Searles, W. H., and H. C. Ives. Field Engineering Handbook, John Wiley & Sons, Inc., New York, 21st ed., 1945, v. 1, 366 pp.; v. 2, 403 pp.

56. Shklyarskiy, F. N. Odin iz Sposobov Opredeleniya Zapasov Gnezdovykh Mestorozhdeniy (A Method of Computing Reserves for Nestlike Ore Bodies). Gorn. Zhur. (Mining Journal), Nos. 4-5, 1921, pp. 158-160.

57. Smirnov, V. I., A. P. Prokof'yev, and others. Podschet Zapasov Mestorozhdenii Poleznykh Iskopaemykx (Reserve Computations of Mineral Deposits). Gosgeoltekhizdat (State Scientific and Technical Publishers for Mining Literature), Moscow, U.S.S.R., 1960, 672 pp.

58. Soderberg, A. Mining Methods and Costs at the Utah Copper Co., Bingham Canyon, Utah. BuMines Inf. Circ. 6234, 1930, 23 pp.

59. Stammberger, Friedrich. Uber Ungenauigkeiten und "erlaubte Fehlergrensen" bei Vorratsberedhnungen (Inexactitude and "Permissible Limits of Error" in the Evaluation of Reserves). Ztschr. für angewandte Geologie, No. 4, 1956, pp. 180-189.

60. Stoddard, A. C. Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz. BuMines Inf. Circ. 6169, 1929, 23 pp.

61. Thomas, R. W. Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz. BuMines Inf. Circ. 6167, 1929, 27 pp.

62. Truscott, S. J. Mine Economics. Rev. and enl. by James Russell. Mining Publications, Ltd., 3d ed., London, 1962, 471 pp.

63. U.S. Atomic Energy Commission. Manual on Ore Reserve Procedures. (For official use only), 1958, 99 pp.

64. Vinogradov, B. C. Znacheniye Metoda Podscheta Zapasov Geologicheskimi Blokami v Usloviyakh Podmoskovskogo Basseina (The Value of Reserve Computations by Geologic Blocks Under Conditions in Moscow Coalfields). Razvedka i Okhrana Nedr (Exploration and Conservation of Mineral Resources), No. 4, 1958, pp. 14-15.

65. Wolff, J. W., E. L. Derby, and W. A. Cole. Sampling and Estimating Lake Superior Iron Ores. Trans. AIME, v. 72, 1925, pp. 641-652.

66. Yakzhin, A. A. Oprobovaniye i Podschet Tverdykh Poleznykh Iskopaemykh (Sampling and Reserve Estimates for Solid Mineral Deposits). Gosgeolizdat (State Publishers for Geologic Literature), Moscow, U.S.S.R., 1954, 296 pp.

67. Yufa, B. Ya. Metodika Otsenki Sluchainykh Pogreshnostey Analizov Pri Podschete Zapasov Mineral'nogo Syr'ya (Methods of Appraisal Random Errors of Chemical Analyses When Computing Reserves of Mineral Raw Materials). Razvedka i Okhrana Nedr (Exploration and Conservation of Mineral Resources), No. 12, 1960, pp. 7-14.

68. Zhuravskiy, A. M. Obshchiye Metody Podscheta Zapasov Rudnykh Mestorozhdeniy (General Methods for Computing Ore Reserves). Trudy Glavnogo Geologo-Razvedochnogo Upravleniya V.S.N. Kh. S.S.S.R. (Transactions ot the Geological and Prospecting Service of U.S.S.R.). Issue 116, Moscow-Leningrad, 1931, 40 pp. (Summary in English.)

69. _____. 1. Ob Odnom Obobshchenii formuly V. I. Baumana (On Generalized Bauman's Formula). 2. K Metodike Podscheta Zapasov Rudnykh Tel (Method of Ore Reserve Computations). Trudy Glavnogo Geologo-Razvedochnogo Uprableniya V.S.N. Kh. S.S.S.R. (Transactions of the Geological and Prospecting Service of U.S.S.R.) Issue 201, Moscow-Leningrad, 1932, 25 pp.

APPENDIX A. - ENGLISH AND METRIC SYSTEMS AND CONVERSION FACTORS

English (standard) system		Metric system	
Units	Conversion factor to metric system	Units	Conversion factor to English system
Linear Measure			
Foot....................	0.3048 m	Meter...........	3.2808 ft
Yard (3 ft).............	0.9144 m	Kilometer.......	0.6214 mile
Mile (5,280 ft).........	1.6093 km		
Land or Area Measures			
Square foot.............	0.0929 sq m	Square meter....	10.764 sq ft
Square yard (9 sq ft)...	0.8361 sq m	Square kilometer	0.3861 sq mi
Acre (43,560 sq ft).....	0.4047 hectare	Hectare.........	2.4710 acres
Square mile (640 acres).	2.5900 sq km		
Volume Measure			
Cubic feet..............	0.0283 cu m	Cubic meter.....	35.314 cu ft 1.3079 cu yd
Cubic yard..............	0.7646 cu m		
Weight Measure			
Ounce[1]..................	28.350 g	Gram............	0.0353 oz 0.0322 oz (troy)
Pound...................	0.4536 kg		
Ounce (troy)............	31.103 g	Kilogram........	2.2046 lb
Pound (troy)............	0.3732 kg	Metric ton......	1.1023 short ton 0.9842 long ton
Ton, short (2,000 lb)...	0.9072 metric ton		
Ton, long (2,240 lb)....	1.0160 metric ton		

[1] Avoirdupois weights when not noted.

Source: Peele, Mining Engineers' Handbook, v. 2, pp. 45-49.

APPENDIX B. - USAGE OF VARIOUS METHODS FOR RESERVE COMPUTATIONS FOR SOLID MINERAL DEPOSITS IN U.S.S.R.
(percentage of totals)

TABLE B-1. - Computation by type of solid mineral deposits[1]

Method	Ore deposits	Nonmetallic deposits	Coal and oil shale deposits
Geological blocks	37	46	69
Mining blocks	12	-	-
Cross sections	48	37	1
Isolines	-	2	-
Polygons	2	14	30
Triangles	1	1	-
Total	100	100	100

[1] As considered by the All Union Committee of Mineral Reserves, U.S.S.R., for 1941-61.

Source: Reference 57, table 22, p. 205.

TABLE B-2. - Solid mineral deposits by selected years

Method	1941-47	1951	1954
Geological blocks	16	12	36
Mining blocks	34	24	15
Cross sections	14	45	46
Polygons	22.5	15	2
Triangles	2.5	1	1
Rectangular	-	1	-
Others	11.0	2	-
Total	100	100	100

Source: Reference 57, table 23, p. 206.

APPENDIX C. - A COMPARISON OF RESERVE COMPUTATIONS MADE BY VARIOUS METHODS (U.S.S.R.)

TABLE C-1. - Polymetal deposit in Altai, U.S.S.R.[1]

Methods of computations	Reserves, percent					
	Ore	Copper	Lead	Zinc	Gold	Silver
Average factors and area:						
Thickness-weighted average...........	100	100	100	100	100	100
Average factors and area (arithmetic average)..................	100	85	105	102	91	90
Cross sections.........................	91	97	77	87	90	99
Polygon................................	99	104	86	94	99	106
Triangles..............................	95	104	91	94	94	90
Triangles modified by the areas of influence for each working............	95	99	79	87	94	94

[1] Data for 26 holes--3 holes crossed high-grade lead ore of narrow width.

Source: Reference 46, table 3, p. 29.

TABLE C-2. - Bauxite deposit in Tichvin district, U.S.S.R.[1]

Average factors and area (arithmetic average):	
Index..	100
Cross sections..	103.1
Polygons..	99.3
Triangles...	97.2

[1] Data for 3 bauxite deposits explored by 41 holes.

Source: Reference 46, table 4, p. 29.

APPENDIX D. - FORMULAS

Page

Main Elements

1. Relationship between true, horizontal, and vertical thicknesses.

$$t_{tr} = t_h \sin\beta = t_v \cos\beta$$ 6

2. True thickness - correction, when $\alpha = 0$.

$$t_{tr} = t_{ap} \sin(\beta + \theta)$$ 8

3. Horizontal thickness - correction, when $\alpha = 0$.

$$t_h = t_{ap} \frac{\sin(\beta + \theta)}{\sin\beta}$$ 8

4. Vertical thickness - correction, when $\alpha = 0$.

$$t_v = t_{ap} \frac{\sin(\beta + \theta)}{\cos\beta}$$ 8

5. General case - true thickness

$$t_{tr} = t_{ap} (\cos\alpha \sin\beta \cos\theta + \sin\beta \sin\theta)$$ 8

6. General case - horizontal thickness

$$t_h = t_{ap} (\cos\alpha \cos\theta - \cot\beta \sin\theta)$$ 8

7. General case - vertical thickness

$$t_v = t_{ap} \cos\theta (\cos\alpha \tan\beta + \tan\theta)$$ 8

Rule of Gradual Changes

8. $$AC = \frac{(t_c - t_1)}{(t_2 - t_1)} AB$$ 10

9. $$tc = \frac{AC}{AB} (t_2 - t_1) + t_1$$ 10

Computations

10. Average thickness

$$t_{av} = \frac{t_1 + t_2 + t_3 + \ldots + t_n}{n}$$ 23

Page

11. Area - Trapezoid formula

$$S = \frac{(a + b)}{2} h$$ 25

12. Trapezoidal rule

$$S = h \left[\frac{(a_1 + a_n)}{2} + a_2 + a_3 + \ldots + a_{n-1} \right]$$ 26

13. Simpson's rule

$$S = \frac{1}{3} h \, (a_1 + 2 \Sigma a_{odd} + 4 \Sigma a_{even} + a_n)$$ 26

14. Volume for a block

$$V = LBT$$ 27

15. $$V = St_{av}$$ 27

16. Volume for the entire body

$$V = V_1 + V_2 + V_3 + \ldots + V_n = t_1 S_1 + t_2 S_2 + t_3 S_3 + \ldots + t_n S_n$$ 27

17. Weight - ore tonnage

$$Q = \frac{V}{F} \text{ and } Q = Vf$$ 27

18. $$Q = Q_1 + Q_2 + Q_3 + \ldots + Q_n = V_1 f_1 + V_2 f_2 + V_3 f_3 + \ldots + V_n f_n$$ 28

19. $$Q = VD$$ 28

20. $$D_{nat} = \frac{D_m (1 - P_o)}{(1 - M_o)}$$ 28

21. $$F_{s.t.} = \frac{2{,}000}{62.5 \, D} \text{ ft}^3/\text{s.t. or } F_{L.t.} = \frac{2{,}240}{62.5 \, D} \text{ ft}^3/\text{L.t.}$$ 29

22. $$f = \frac{2{,}000}{F_{s.t.}} \text{ or } \frac{2{,}240}{F_{L.t.}} \text{ or } 62.5 \, D$$ 29

23. $$Q = V_1 D_1 + V_2 D_2 + V_3 D_3 + \ldots + V_n D_n$$ 29

Grade

24. Arithmetic average

$$c_{av} = \frac{c_1 + c_2 + c_3 + \ldots + c_n}{n}$$ 30

Page

25. Thickness - Weighted average

$$c_{av} = \frac{c_1 t_1 + c_2 t_2 + c_3 t_3 + \dots + c_n t_n}{t_1 + t_2 + t_3 + \dots + t_n}$$ 30

26. Area - Weighted average

$$c_{av} = \frac{c_1 S_1 + c_2 S_2 + c_3 S_3 + \dots + c_n S_n}{S_1 + S_2 + S_3 + \dots + S_n}$$ 30

27. Volumetric average

$$c_{av} = \frac{c_1 V_1 + c_2 V_2 + c_3 V_3 + \dots + c_n V_n}{V_1 + V_2 + V_3 + \dots + V_n}$$ 30

28. Gravimetric average

$$c_{av} = \frac{c_1 Q_1 + c_2 Q_2 + c_3 Q_3 + \dots + c_n Q_n}{Q_1 + Q_2 + Q_3 + \dots + Q_n}$$ 30

29a. Weight-metal tonnage

$$P = Q\, c_{av}$$ 30

29b.

$$c_{av} = \frac{P}{Q}$$ 30

30. Correction factor for grade

$$E = \frac{C_y}{C_x}$$ 35

Errors

31. Errors for the entire body

$$M_{av} = \frac{M_1 + M_2 + M_3 + \dots + M_n}{N}$$ 36

32.

$$M_{av} = \frac{M_1 P_1 + M_2 P_2 + \dots + M_n P_n}{NP}$$ 36

Mining Blocks Method

33.

$$t_{av} = \frac{t_1 + t_2 + t_3 + t_4}{4}$$ 48

34.

$$c_{av} = \frac{c_1 + c_2 + c_3 + c_4}{4}$$ 48

		Page
35.	$t_{av} = \dfrac{t_1 L_1 + t_2 L_2 + t_3 L_3 + t_4 L_4}{L_1 + L_2 + L_3 + L_4}$	48
36.	$c_{av} = \dfrac{c_1 L_1 + c_2 L_2 + c_3 L_3 + c_4 L_4}{L_1 + L_2 + L_3 + L_4}$	48
37.	$t_{av} = \dfrac{t_1 s_1 + t_2 s_2 + t_3 s_3 + t_4 s_4}{s_1 + s_2 + s_3 + s_4}$	48
38.	$t_{av} = \dfrac{t_1 L_1 + t_2 L_2}{L_1 + L_2}$	51
39.	$c_{av} = \dfrac{c_1 L_1 + c_2 L_2}{L_1 + L_2}$	51
40.	$t_{av} = \dfrac{3t_1 + t_2}{4}$	51
41.	$c_{av} = \dfrac{3c_1 + c_2}{4}$	51
42.	$t_{av} = \dfrac{t_1 + t_2 + \ldots + t_n + t^1_1 + t^1_2 + \ldots + t^1_m}{n + m}$	53
43.	$c_{av} = \dfrac{c_1 + c_2 + \ldots + c_n + c^1_1 + c^1_2 + \ldots + c^1_m}{n + m}$	53
44.	$t_{av} = \dfrac{t_1 + t_2 + \ldots + t_n + t^1_1}{n + 1}$	53
45.	$c_{av} = \dfrac{c_1 + c_2 + \ldots + c_n + c^1_1}{n + 1}$	53

Cross Sections Method

46. Mean-area formula

$$V = \frac{(S_1 + S_2)}{2} L$$ 56

47. End-area formula (equal distances between sections)

$$V = (S_1 + 2S_2 + 2S_3 + \ldots + S_n) \frac{L}{2}$$ 58

48. Volume for entire body (unequal distances between sections)

$$V = \frac{(S_1 + S_2)}{2} L_1 + \frac{(S_2 + S_3)}{2} L_2 + \ldots + \frac{(S_{n-1} + S_n)}{2} L_n$$ 58

Page

Linear Method

60. Ore tonnage (based on linear ore reserves)

$$Q = q_L A$$ 68

61. Metal tonnage (based on linear metal reserves)

$$P = p_L A$$ 68

62. Ore tonnage (based on ore reserves per cubic unit)

$$Q = q_v V$$ 69

63. Metal tonnage (based on metal reserves per cubic unit)

$$P = p_v V$$ 69

Method of Isolines

64. Volume

$$V_2 = h \frac{S_1 + (S_2{}^{1} + S_2{}^{11})}{2}$$ 74

65.

$$V_3 = \frac{h}{2}\left[(S_2{}^{1} + S_2{}^{11}) + (S_3{}^{1} - S_3{}^{11})\right]$$ 74

66. Grade

$$c_{av} = \frac{c_o A_o + \frac{c}{2}(A_o + 2A_1 + 2A_2 + \ldots + A_n)}{A_o}$$ 74

Method of triangles

67. For triangular prism - volume

$$V = \frac{1}{3}(t_1 + t_2 + t_3)\ S$$ 77

68. For triangular prism - grade

$$c_{av} = \frac{c_1 + c_2 + c_3}{3}$$ 77

69.

$$c_{av} = \frac{c_1 t_1 + c_2 t_2 + c_3 t_3}{t_1 + t_2 + t_3}$$ 77

70. Ore tonnage

$$Q = \frac{V}{F} \text{ or } \frac{(q_1 + q_2 + q_3)}{3}\ S$$ 77

Page

71. Metal tonnage

$$P = Qc_{av} \text{ or } \frac{(q_1c_1 + q_2c_2 + q_3c_3)}{3} S \text{ or } \frac{(p_1 + p_2 + p_3)}{3} S$$ 77

72. Number of connecting lines

$$N_L = 3n_1 + 2n_2 + 3, \text{ or } N_L = 3(n_1 - 1) + 2n_2$$ 79

73. Number of triangles

$$N_{tr} = 2n_1 + n_2 - 2 = 2(n_1 - 1) + n_2$$ 79

74. The relative error for rectangular prism

$$\Delta V = V_1 - V_2 = \frac{1}{6}(t_1+2t_2+t_3+2t_4) S - \frac{1}{6}(2t_1+t_2+2t_3+t_4) S$$

$$\text{or } \Delta V = \pm \frac{S}{6}(t_1-t_2+t_3-t_4)$$ 82

75. Condition for precise volume computations in rectangular prism

$$t_1 + t_3 = t_2 + t_4$$ 82

76. General case - average thickness

$$t_{av} = \frac{4t_1 + t_2 + t_3}{6}$$ 83

77. Case of $t_3 = 0$

$$t_{av} = \frac{3t_1 + t_2}{6}$$ 83

78. Case of t_2 and $t_3 = 0$

$$t_{av} = \frac{t_1}{3}$$ 83

79. Volume for the entire body ($s = \frac{S}{N}$)

$$V = \frac{1}{3} s \sum_{i=1}^{N} tk$$ 83

80. Ore reserves for the entire body ($s = \frac{S}{N}$)

$$Q = \frac{1}{3} s \sum_{i=1}^{N} tfk$$ 83

	Page
81. Metal reserves for the entire body ($s = \frac{S}{N}$)	
$$P = \frac{1}{3} s \sum_{i=1}^{N} tfck$$	83
82. Metal tonnage	
$$P = c_1 q_1 + c_2 q_2 + c_3 q_3 + \ldots + c_n q_n$$	87
83. Square net of workings - one block	
$$V = \frac{(t_1 + t_2 + t_3 + t_4)}{4} s$$	88
84. Square net of workings for the entire body ($s = \frac{S}{N}$)	
$$V = \frac{t_1 + t_2 + t_3 + \ldots + t_n}{N} S = (t_1 + t_2 + t_3 + \ldots + t_n) s$$	88
85. Total average relative error	
$$M_{av} = \sqrt{m_{te}^2 + m_a^2 + m_g^2}$$	92

APPENDIX E. - GLOSSARY

Selected terms used in this report are included in the following definitions.

Area reserves. - Reserves computed for a certain area.

Arithmetic average or mean. - Simple average of a suite of quantities, measurements, analyses, etc.; sum of a suite divided on the number of quantities.

Block. - A unit of mineral body delineated by various principles of interpretation of exploration data; varies with the method of computations.

Inside perimeter. - The portion of a mineral deposit delineated by outlying mine workings.

Linear metal reserves. - Metal reserves for an area unit, such as square foot foot and square meter (product of linear ore reserves and average grade), in tons or other weight units.

Linear ore reserves. - Ore reserves computed for an area unit, such as square foot or square meter, in tons or other weight units.

Mine workings. - All surface and underground exploration, development, and exploitation exposures of a mineral body; drilling included.

Ore. - A mineral substance that can be mined at present at profit to the operator or to the benefit of the nation.[1]

Outside perimeter. - Portion of a mineral deposit extended beyond the outlying mine workings; delineated according to geological evidence or certain principles.

Parameters of a mineral body. - A series of physical and chemical constants which express this body.

Reserves. - Mineral material considered exploitable under existing conditions; including cost, price, technology, and local circumstances (7).

Resources. - Reserves plus potential raw material; includes marginal, submarginal, and latent categories (7).

Section reserves. - Reserves along a section--one unit of length wide.

Segment. - A large portion of a mineral body.

Underground mine workings. - Exploration and development shafts, adit, drifts, crosscuts, raises, and winzes; drilling excluded.

Unit volume reserves. - Reserves computed for one unit of volume, such as cubic foot, cubic yard, or cubic meter.

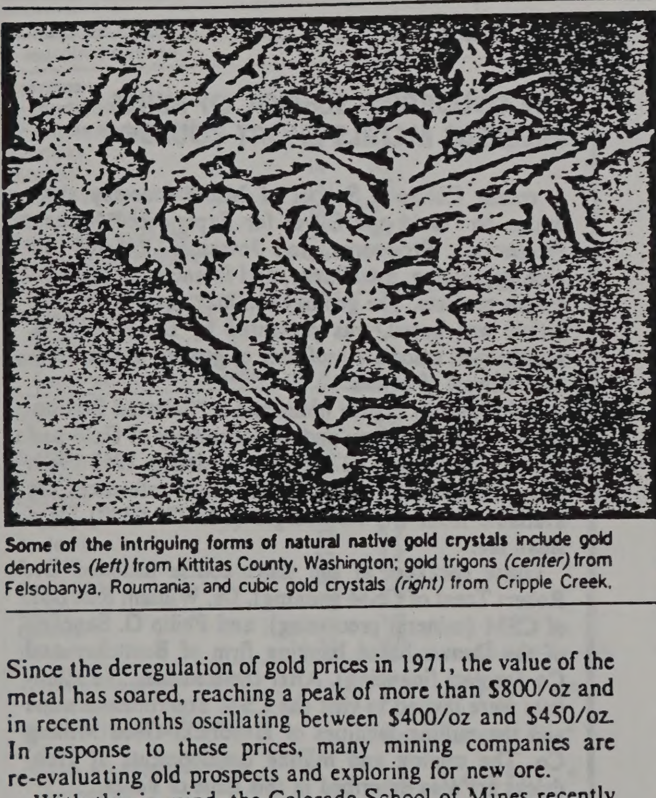
[1] Definition approved by the United Nations Educational, Scientific and Cultural Organizations.

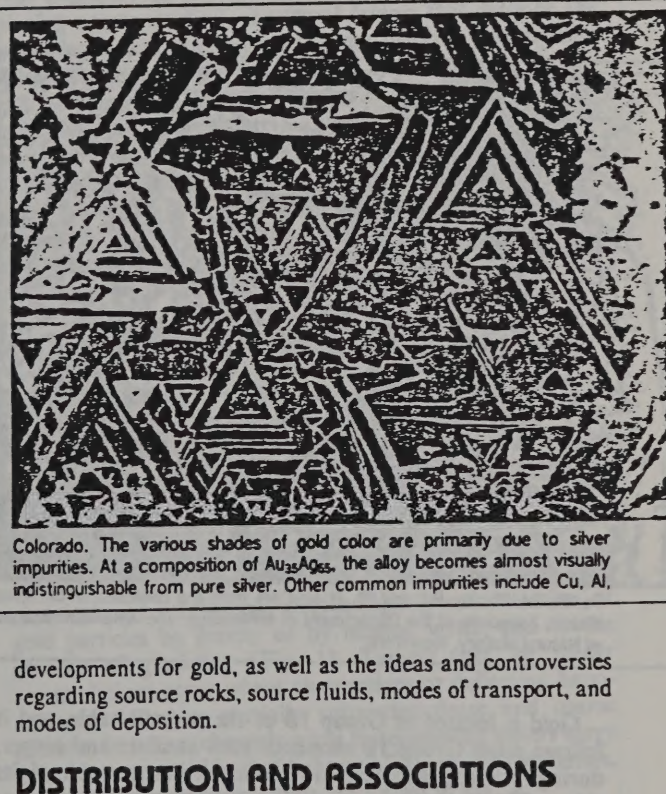
THIS PAGE
INTENTIONALLY
BLANK

Bob Lewis:
Handouts

GOLD GEOLOGY BASICS

Alvin Lewis, Associate Editor





Some of the intriguing forms of natural native gold crystals include gold dendrites *(left)* from Kittitas County, Washington; gold trigons *(center)* from Felsobanya, Roumania; and cubic gold crystals *(right)* from Cripple Creek, Colorado. The various shades of gold color are primarily due to silver impurities. At a composition of $Au_{35}Ag_{65}$, the alloy becomes almost visually indistinguishable from pure silver. Other common impurities include Cu, Al,

Since the deregulation of gold prices in 1971, the value of the metal has soared, reaching a peak of more than $800/oz and in recent months oscillating between $400/oz and $450/oz. In response to these prices, many mining companies are re-evaluating old prospects and exploring for new ore.

With this in mind, the Colorado School of Mines recently held its annual short course in gold and silver technology, which included a section on gold geology for those new to gold prospecting (see box). The following article is based on the geology seminar, and offers an introduction to the major types of gold deposits, requisites for their formation, and descriptions of several significant worldwide occurrences.

A second E&MJ article, based on an upcoming Penn State seminar, is planned for the July issue of the magazine and will cover in more detail current geochemical exploration developments for gold, as well as the ideas and controversies regarding source rocks, source fluids, modes of transport, and modes of deposition.

DISTRIBUTION AND ASSOCIATIONS

Gold is widely distributed in the Earth's crust and its oceans, but only rarely does it occur in concentrations great enough to permit economic recovery. Crustal rocks contain an average of only 5 ppb of gold, and some more auriferous crustal rock types average up to 10-12 ppb (Table 1). To produce an orebody, these low-grade "source rocks" must be geologically reworked under special conditions that concentrate the gold values by some 3,500 times. Such conditions, however, are rare in geologic space and time.

FOR THOSE NEW TO GOLD PROSPECTING, HERE'S AN INTRODUCTION TO AURIFEROUS PLACER AND HYDROTHERMAL ORE DEPOSITS

Fe, Bi, Sn, Pb, Zn, Pd, and Pt. Photos are from the collection of Dr. Julius Weber, Associate of the Department of Mineralogy, The American Museum of Natural History, New York.

Gold is located in Group 1B of the periodic table, and it follows other Group 1B elements, such as silver and copper, during primary geochemical processes. As a result of its siderophile characteristics, gold concentrates in residual fluids and subsequent metallic or sulphide phases rather than in the earlier silicate crystals of cooling magmas. In hydrothermal deposits, gold is associated with mercury, bismuth, antimony, arsenic, selenium, tellurium, and thallium, as well as silver and copper. In magmatic deposits, gold is associated with the platinum group metals (PGMs).

Gold usually occurs in its native form or as gold-silver electrums or tellurides. It often substitutes for other chemically similar elements or forms tiny inclusions in the more common minerals, such as in pyrite, arsenopyrite, chalcopyrite, stibnite (Sb_2S_3), orpiment (As_2S_3), and realgar (AsS).

THE COLORADO SCHOOL OF MINES PRECIOUS METALS SHORT COURSE

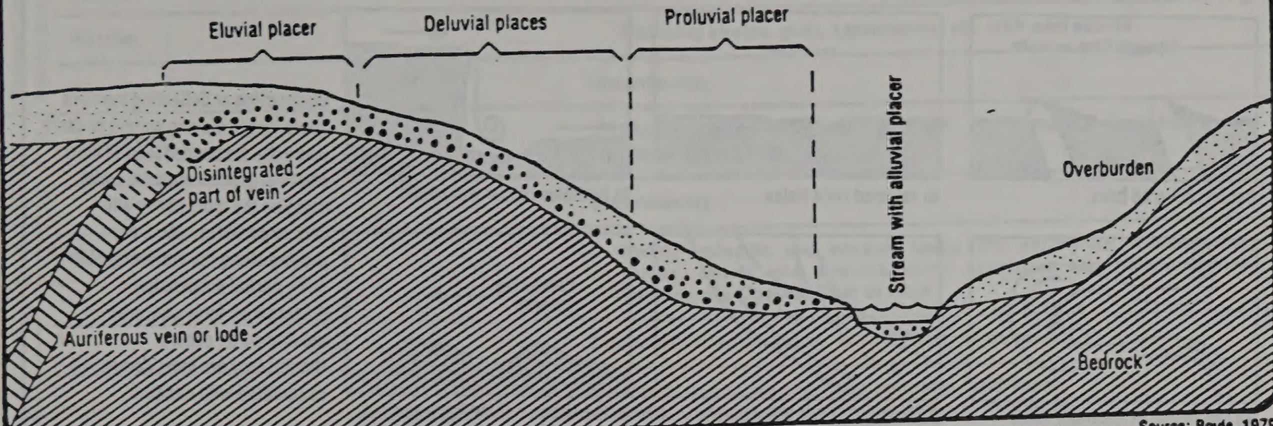
A recent Colorado School of Mines four-day short course on "Gold and Silver for Experienced Miners, Prospectors, Processors, and Investors" covered the geology, mining, metallurgy, and financing of precious metals projects. The geology seminar, upon which this article was based, was presented by Dr. Samuel B. Romberger, associate professor of geology at the Colorado School of Mines.

Gold geology course notes and a "Selected Bibliography on the Geology and Geochemistry of Gold and Silver," containing 927 references dating 1950-1981, was distributed to all attendees. These references are available from Dr. Romberger (303-279-0300) for a small fee.

Other speakers at the short course included Dr. Robert Trent of CSM (mining), Dr. William Rex Bull of CSM (mineral processing), and Philip O. Sancken of the Denver-based banking firm of Boettcher and Co. (project financing). After the seminars, the attendees were invited to visit the CSM experimental mine and the milling facilities of Hendricks-Good Milling Co. The mining and milling developments at Hendricks are also described in this issue of E&MJ.

The Colorado School of Mines plans to hold two "gold and silver" short courses annually; the above course is held every October, and a more basic introductory course is held each April. Dan Witkowsky, of the US Bureau of Mines in Denver, is the organizer-director of both courses. For information about either course, contact the CSM Dept. of Continuing Education at 303-279-0300.

Fig. 1—Outcrop of a gold-quartz vein supplying material to form eluvial and alluvial placers



Source: Boyle, 1979.

Table 1—Gold content of selected igneous, metamorphic, and sedimentary rocks

Rock type	No. of analyses	Av.	Au (ppb) Range	Refer-ence
Felsic ig. & mm. rocks	11	2.4	0.6- 4.2	(2)
Mafic ig. & mm. rocks	10	2.6	2.0- 3.2	"
Mid-Atlantic Ridge basalts	3	10	7-13	"
Skaergaard Intrusion: marginal gabbro		4.6		(3)
Granites	33	3.2		(4)
Granodiorites	8	4.0		"
Rhyolites and trachites	14	5.4		"
Syenites	8	4.4		"
Diorites	14	3.5		"
Altai-Sayan gabbroids	14	6.4		"
Intrusive basalts	19	10		"
Gabbroids, average	33	8.7		"
Basalts and andesites	29	6.5		"
Ultramafic rocks	27	9.4		
Tholeiitic basalt (Mauna Loa, Hawaii)	3	4.2		(5)
Mid-Atlantic Ridge and East Pacific Rise basalts	4	0.3		"
Granite	13	2.8		(6)
Syenite	9	2.5		"
Gabbro	23	5.4		"
Dunite	25	8.2		"
Rhyolite	2	12		"
Trachyte	12	6.5		"
Andesite	13	5.2		"
Basalt	11	3.2		"
Sandstone	24	7.5		"
Shale	8	3.9		"
Gneiss	1	1.8		"
Schist	12	5.0		
Gabbro (Southern Calif. batholith)	6	4.7	0.7-11.6	(7)
Tonalite (Southern Calif. batholith)	3	5.0	1.7-11.3	"
Quartz monzonite (Idaho batholith)	7	1.0	0.4- 2.7	"
Low-K_2O basalt (Oceanic ridges)	7	1.2	0.5- 2.0	"
Alkali basalt (Oceanic ridges)	10	0.8	0.2- 1.8	"
Diabase (Southeast Missouri)	18	2.8	1.1- 7.0	"
Basalt (Snake River, Idaho)	19	0.5	0.1- 1.3	"
Dacite (W. Cascades, Ore.)	6	1.1	0.6- 1.4	"
Andesite (W. Cascades, Ore.)	9	3.4	0.5-10.7	"
Basalt (W. Cascades, Ore.)	4	4.6	3.3- 5.3	

Source: Stephenson and Ehmann (1971) [10]

The geochemical conditions required to form auriferous ore deposits are highly complex and are being studied by research scientists. To understand these deposits and predict their occurrence, geologists and geochemists try to decipher the sources of the mineralizing solutions (i.e., magmatic, meteoric, or metamorphic), sources of the precious metals (i.e., magmas or pre-existing rocks through which the mineralizing fluids migrate), mechanisms of transport (i.e., solution chemistry as affected by temperature, pressure, complexing agents, acidity), structural factors that describe the plumbing system (e.g., calderas, joints, breccias, permeability of sandstones or conglomerates, etc.), mechanisms of deposition (e.g., cooling, boiling, changes in pH, changes in the oxidation state, dilution, etc.), and the geology at the depositional environment.

The most important gold deposits occur either as *placers* or as *hydrothermal ores*. Gold can also be economically recovered as a by-product of some magmatic deposits, such as in the nickeliferous Sudbury Basin of northern Ontario, but these types of occurrences will not be described here.

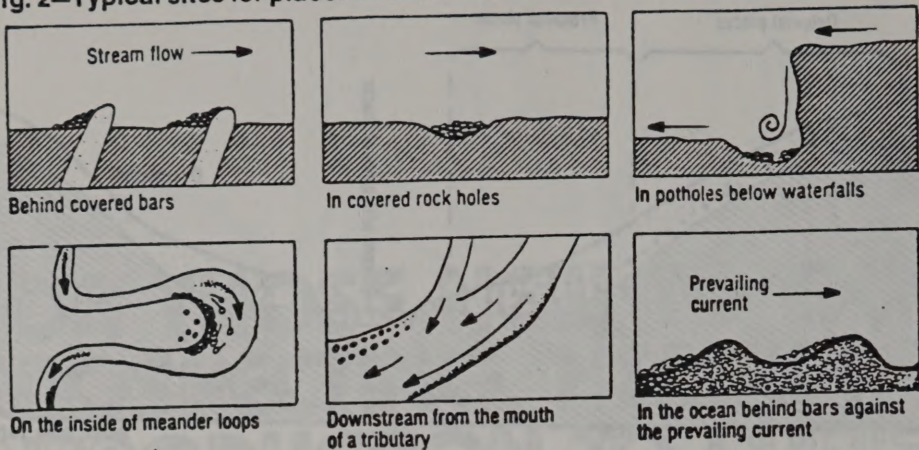
GOLD PLACER DEPOSITS

Most of the world's economically recoverable gold is in placers, an alluvial or sometimes eluvial type of deposit in which gold has been concentrated through physical processes because of its high density and weather-resistant characteristics. Several prerequisites are necessary to form placer deposits, including: 1) a source for the gold, which could be gold-quartz veins, gold-bearing sulphide deposits, auriferous conglomerates, or previous placers; 2) a long period of deep chemical and mechanical weathering; 3) concentration of gold particles by gravity or by moving water; and 4) the absence of glaciation (Fig. 1). Typical sites for placer accumulations occur where obstructions or deflecting barriers trap rolling or partially suspended dense and coarse particles in stream channels while faster-moving waters carry away the suspended load of light and finer-grained materials. Some of the more typical sites are shown in Fig. 2.

Because of its higher density relative to other river sediments, gold particles collect at these obstructions or where they suddenly enter a lower-energy environment. For these reasons, the richest and coarsest gold fraction often occurs with the coarser sedimentary material, in the upper reaches of the stream valley, and/or at the bottom of steep gradients. The finer gold typically occurs downstream at lower gradients, with sandier sediments.

In some areas, subsequent depositional processes have

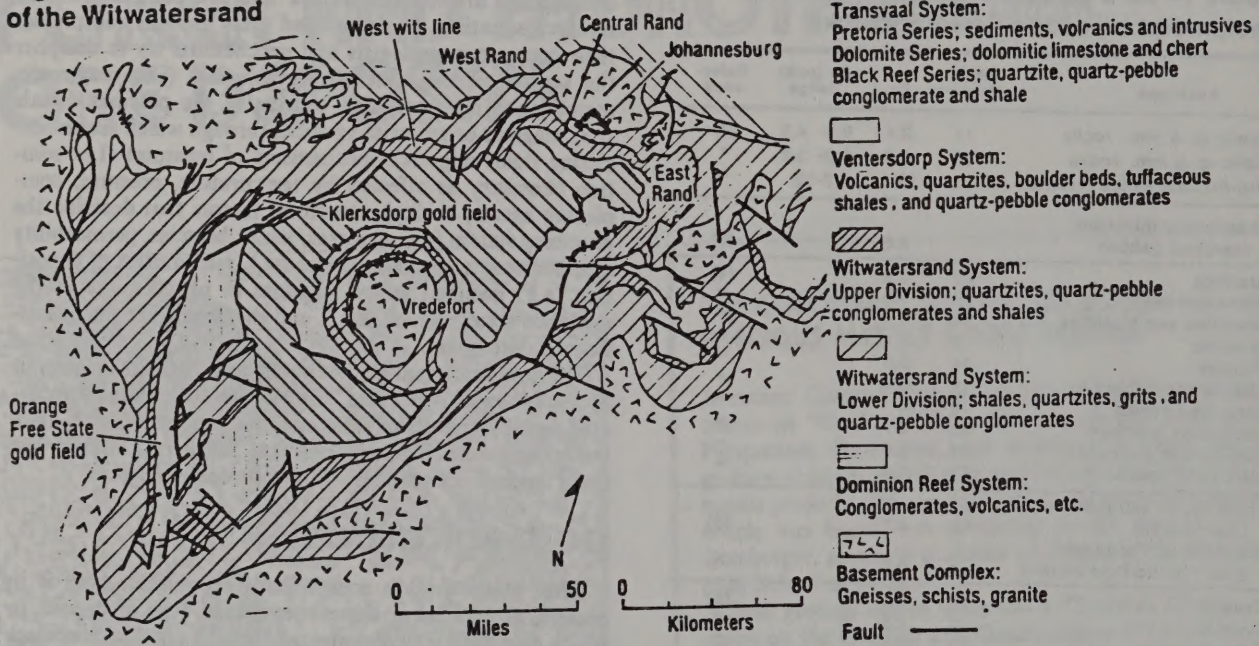
Fig. 2—Typical sites for placer accumulations

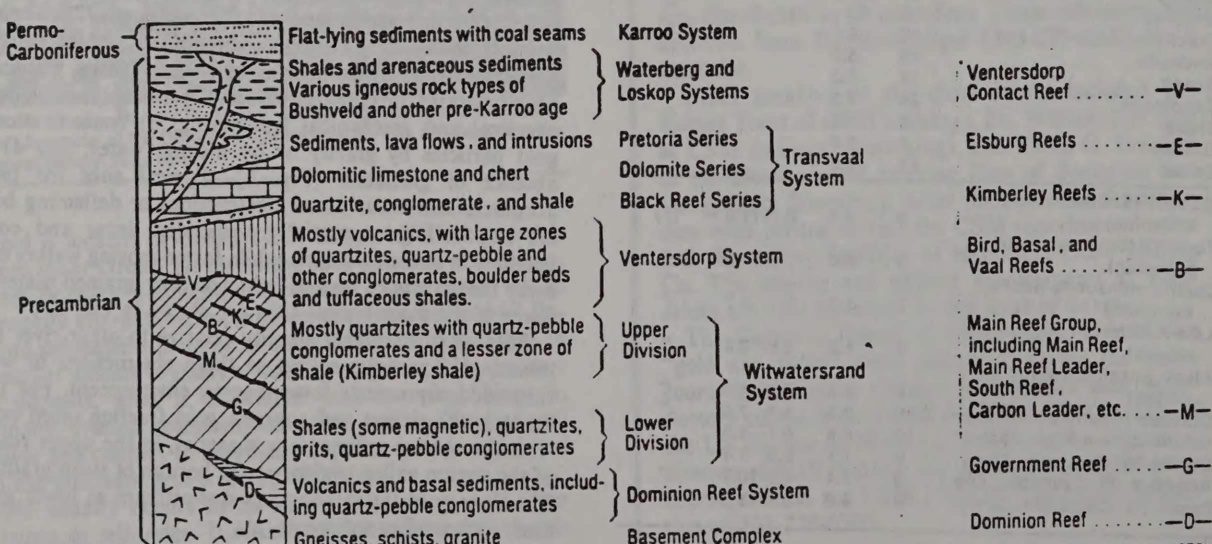


Obstructing or deflecting barriers allow faster-moving waters to carry away the suspended load of light and fine-grained material while trapping the more dense and coarse particles, which are moving along the bottom by rolling or by partial suspension. Placers may form wherever moving water occurs, though they are most commonly associated with streams.

From Skinner, 1969.

Fig. 3—Generalized geological map and geological column of the Witwatersrand





Source: Boyle, 1979.

Table 2—Generalized geological column, Witwatersrand Basin, South Africa

	System		Lithology and remarks
Permo-Carboniferous	Karroo		Flat-lying shales, grits, sandstones, etc. with coal seams
			Unconformity
Precambrian	Transvaal		Shales, quartzites, dolomite, chert, conglomerate, lava flows, etc. Black Reef at base
			Unconformity
	Ventersdorp		Mainly andesitic and basaltic lavas with porphyries, felsites, tuffs, shales, cherts, and breccia, some quartzites and conglomerates. Ventersdorp Contact Reef at base
			Conformity in some areas; unconformity in others
	Witwatersrand	Upper Division	Mostly quartzites and conglomerates, and minor shale. Contains the auriferous Elsburg, Kimberley, Bird, and Main Reef groups
			Conformity in places; unconformity in others
		Lower Division	Amygdaloidal lava at top, shales, quartzites, grits, conglomerate. Contains the auriferous Government Reef and other gold-bearing reefs near the middle of the division
			Unconformity
	Dominion Reef		Conglomerate, quartzite, grit, arkose, amygdaloidal andesites, felsites, and rhyolite. Auriferous Dominion Reef at base
			Great unconformity
	Swaziland (Basement rocks)		Amphibolites, actinolite, and hornblende schists, talc schists, serpentinites, gneisses, and intrusive granitic rocks

Table 3—Generalized stratigraphical column for the Witwatersrand System in the Central Rand

Division and total thickness (ft)	Series and total thickness (ft)	Main stratigrapical marker horizons	Thickness of marker horizons(ft)
Upper division Witwatersrand System (9,400) Mainly quartzites and conglomerates with minor shale	Kimberley-Elsburg Series 6,100	Elsburg Reefs	1,500
		Kimberley Reefs	600
		Kimberley Shales	550
	Main-Bird Series 3,300	Bird Reef Marker	100
		Bird Reefs	300
		Main Reef Group	100
Lower division Witwatersrand System (15,200) Mainly shales, quartzites, grits, and conglomerates	Jeppestown Series 3,800	Jeppestown	100
		Amygdaloid	—
	Government Reef Series 6,100	Government Reef	5
		Coronation Reef	5
		Coronation (or West Rand) Shales	500
		Promise Reef	5
	Hospital Hill Series 5,300	Hospital Hall Quartzites	1,300
		Contorted Bed	150
		Speckled Bed	5
		Ripple-marked Quartzites	20
		Water Tower Slates	800
		Orange Grove Quartzites	550

Source: Brock and Pretorius (1964b).[8]

buried the placer deposits, rendering them more difficult to find. For example, gold placers are buried under volcanic rocks in California and Australia, under glacial deposits in eastern Canada, under eolian deposits in Australia, and under talus deposits, alluvial sands and gravels, and marine and lacustrine deposits in other areas.

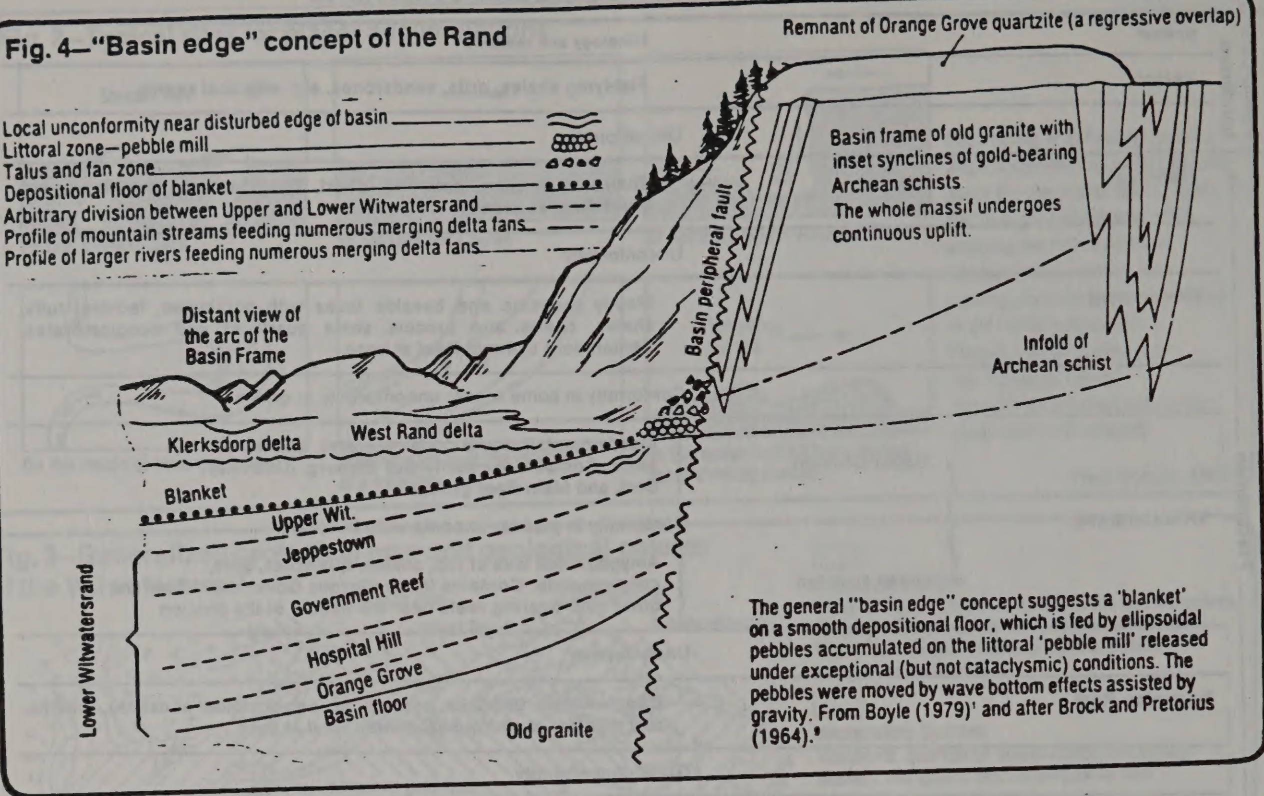
The largest known gold deposits are the "paleoplacers," also known as quartz pebble conglomerates. These deposits have produced about 60% of the world's gold in the past 50 years and are often associated with uranium, thorium, and the rare earth elements. The best known examples are in the Witwatersrand in South Africa, Jacobina in Brazil, and Elliot Lake and Blind River in Canada. Less well-known paleoplacers occur in Ghana, Rhodesia, Tanzania, Zaire, Finland, Australia, and the USSR.

All of the productive quartz-pebble conglomerates are Precambrian in age and lie within a sedimentary sequence overlying the unconformity between the Archean basement and Lower Proterozoic rocks. Mineralization is often as lenses, stringers, or leads; and though only a few feet thick, they can extend from tens to hundreds of miles. The pebbles are predominantly quartz with some chert, porphyry, and slate, and are often rounded, sorted, and abraded. The fine-grained matrix is composed of quartz, sericite, chlorite, chloritoid, and accessory minerals.

THE WITWATERSRAND BASIN

Over 1 billion oz of gold, 100 million oz of silver, and significant quantities of uranium and platinum group metals have been recovered from the Witwatersrand of South Africa, well known as the world's largest supplier of gold (30 million oz of gold produced each year). Discovered in 1886,

Fig. 4—"Basin edge" concept of the Rand



The general "basin edge" concept suggests a 'blanket' on a smooth depositional floor, which is fed by ellipsoidal pebbles accumulated on the littoral 'pebble mill' released under exceptional (but not cataclysmic) conditions. The pebbles were moved by wave bottom effects assisted by gravity. From Boyle (1979)[1] and after Brock and Pretorius (1964).[6]

the Witwatersrand Basin consists of a synclinal trough with local peripheral faults and cross-fractures with major orebodies along the northeast, east, west, and southern edges of the basin.

The mineralized horizons, which average 0.1-0.3 oz/t gold, overlie the Swaziland System of deformed Precambrian basement metamorphics and are overlain by the Permo-Carboniferous, flat-lying shales and sandstones of the Karoo System (Tables 2 and 3, Fig. 3). Gold mineralization occurs at the bases of four separate Precambrian Systems: the Dominion Reef, Witwatersrand, Ventersdorp, and Transvaal. The most productive conglomerates, however, occur at the upper division of the Witwatersrand System (Main Reef and Bird Reef).

The most important gold occurrences are in matrices of the conglomerates and in banded quartzites, both of which represent sedimentation in the initial stages of a cycle. Gold also occurs in quartzites along unconformity planes, and in carbon seams, quartz veins, and dykes, all of which represent concentration mechanisms during the terminal stages of a sedimentary cycle. Most of the gold is detrital, but there is some evidence of remobilization by solutions during metamorphism.

Numerous hypotheses have been proposed to explain the depositional environment of the basin's sediments and volcanics, but the most recent model focuses on deposition in a closed fault-bounded continental basin, which became less stable with time (Fig. 4). This "basin edge" concept for the mineralization of the "Rand" involves the concentration of gold in fluvial fan or fan-delta deposits where a river emptied into a lake or inland sea. The detrital material is believed to have been constantly reworked in braided channels at the head of the fan and by shallow water currents. The Rand System is now being reinterpreted as an intermountain lake system with its gold reworked into stream channels. This interpretation supplants older models, which presumed a marine setting for the ore paragenesis.

THE JACOBINA GOLD DEPOSITS

A second major "paleoplacer" occurs at Jacobina, Brazil, where ore grading 0.25-0.45 oz/t gold with significant quantities of uranium is found in Lower Proterozoic conglomerates above the Archean unconformity. The origin of these deposits is considered to be a combination of placer formation and later modification by solutions during metamorphism. Some plate tectonic advocates are eager to point out that Brazil may have been close to the Witwatersrand Basin at some time during the Precambrian. The details of the geology of the deposit are described in *The Geochemistry of Gold and its Deposits.*[1]

HYDROTHERMAL GOLD DEPOSITS

Most hydrothermal gold deposits fall into one of two major categories: those associated with Tertiary volcanism or hot spring activity; and those hosted by Precambrian metamorphic rocks.

Hydrothermal gold and Tertiary volcanism. These deposits can be hosted by volcanic rocks (e.g., the San Juan mountain area of Colorado) or by older Paleozoic-Mesozoic sedimentary rocks (e.g., Carlin, Nevada). The important requisites for these deposits are: 1) extensive masses of volcanics, which provide a good potential source of metals; 2) volcanic collapse calderas, to supply the plumbing systems for the migrating hydrothermal fluids; and 3) continuing igneous activity, which supplies the heat to drive the convecting hydrothermal system.

Fluid inclusion and isotopic evidence indicate that the water is often meteoric, with NaCl equivalent salinites usually less than 1%. Typical mineralization models propose rising heated solutions through the convecting system followed by chemical changes of the fluid and subsequent mineral precipitation. The most important chemical changes

include boiling of the hydrothermal solutions, loss of volatiles with subsequent increase in pH, loss of complexing anions with the vapor, and cooling of the fluids.

The San Juan volcanic field of southwest Colorado is one of the most studied and best known mineralized caldera systems. The total volume of the field exceeds 1,000 km^3, with volcanic sequences often exceeding 2 km in thickness. All of the ore deposits in the area are controlled by structures related to calderas, of which 15 have been recognized in the area. For example, in the Silverton-Lake City area, mineralization is localized in fractures radiating outward from the caldera, in breccia pipes localized along concentric ring fractures bounding the caldera, and in the Eureka graben.

In Nevada, mineralization in Tertiary volcanics is controlled by caldera rim fractures (e.g., Bullfrog, Goldfield), by areas of local uplift that may indicate local intrusions at depth (e.g., Tonapah, Goldfield, Comestock, Bodie, Aurora), and by groups of veins and breccia pipes near centers of the volcanic systems (e.g., Silver Peak, Majuba Hill, and Bodie).*

Low-grade disseminated gold deposits in Nevada and Utah also seem to be associated with Tertiary igneous activity, even though mineralization commonly occurs in rocks as old as the Paleozoic. The largest producer from this type of gold deposit is at Carlin, Nev. Other similar deposits are in Cortez and Getchell, Nev.; and the Mercur District in Utah.

About 3.2 million tr oz of gold have been produced from the Silurian-Devonian Roberts Mountain formation at Carlin. Newmont geologists discovered the deposit via geochemical prospecting techniques in 1962, and the deposit has been open-pit mined since 1965. Mineralization is believed to be controlled by Tertiary normal faults and occurs in northwest-trending fractures, subsidiary fractures, and adjacent carbonates. Organic material is always present and can reach up to 8% near unoxidized ore zones.

The contemporary genetic model for Carlin involves deposition of gold from deeply circulated groundwater in limestone, in association with late-Tertiary hot spring activity. Mineralization is accompanied by extensive solution of the host limestone, which resulted in the permeability required for gold deposition. Fluid inclusion and isotopic studies suggest a groundwater source for the solutions, which were heated to 150-200°C before boiling. Some geologists support their "hot spring model" for Carlin deposits by pointing to gold mineralization and similar chemical characteristics around some present-day geothermal areas, such as Steamboat Springs, Nev., and the Broadlands, New Zealand.

Hydrothermal gold associated with precambrian rocks. This second major type of hydrothermal gold deposit includes the districts of the Colorado Front Range Mineral Belt (Boulder County, Cripple Creek, Breckenridge, and Central City) as well as "a more enigmatic group" in older Precambrian terranes, where there is little relationship to Tertiary activity. Examples of the latter type occur in Yellowknife, NWT; Porcupine-Timmins and Kirkland Lake, Ontario; Rhodesia; Kalgoorlie, Western Australia; and Homestake, S.D. The origin of these deposits is uncertain, as are the sources for the gold, the sources for the fluids, and the age of the mineralization.

* The detailed geology of some of these areas is summarized, with references, in the geology handouts provided by Dr. Romberger in the seminar.

REFERENCES

1) Boyle, R.W. (1979), "The Geochemistry of Gold and Its Deposits," *Geological Survey of Canada*, Bulletin 280, 584 pp.
2) DeGrazia, A.R., and L. Haskin (1964), GEOCHIM COSMOCHIM ACTA, vol. 28, pp 559-564.
3) Vincent, E.A., and J.H. Crockett (1960), GEOCHIM COSMOCHIM ACTA, vol. 18, nos. 1-2, pp 130-148.
4) Shcherbakov, Y.G., and G.A. Perezhogin (1964), GEOKHIM, No. 6, pp 518-528.
5) Ehmann, W.D., and P.A. Baedecker (1968), *Origin and Distribution of the Elements*, L.H. Ahrens, ed., pp 301-311.
6) Jones, R.S. (1969), *U.S. Geol. Survey Circ. 610*, 28 pp.
7) Gottfried, D., et al (1969), *Geol. Soc. Am., Abstr. with Programs*, Part 7, Annual Meeting, Atlantic City, N.J., pp 277-278.
8) Brock, B.B., and D.A. Pretorius (1964b), GEOL SOC. SOUTH AFRICA, vol. 1, pp 25-61.
9) Brock, B.B., and D.A. Pretorius (1964a), GEOL SOC. SOUTH AFRICA, vol. 1, pp 549-599.
10) Stephenson, J.F., and W.D. Ehmann (1971), ECON. GEOL., vol. 66, pp 933-939.

H. W. C. PROMMEL

PAUL M. HOPKINS

PRELIMINARY EVALUATION AND SAMPLING OF GOLD PLACER DEPOSITS

BY H. W. C. PROMMEL AND PAUL M. HOPKINS
Registered Mining Geologists and Engineers

INTRODUCTION

Denver Equipment Company is very much aware of the increased interest in exploring gold placer deposits. Experience and practical business sense point out that the people who put their money into risk ventures need to have certain basic facts on which to base their decisions; facts which will reduce the risk and increase the chances of successful operations of the venture.

The expenditure of a small amount of time and money in proper evaluation of a deposit can help predict economic success or failure before extensive financial commitments are made. In addition, proper evaluation frequently can indicate a successful venture where possibly a less thorough or unscientific appraisal would reject a deposit or indicate that development would be a waste of effort. The secret, of course, lies in securing reliable data on which to base intelligent decisions.

Due to the inactivity of gold mining in the U. S. A. during the past two or three decades and the limited production of gold throughout the rest of the world, experienced placer engineering skills become more difficult to secure.

The editors of Deco Trefoil have persuaded Mr. H.W.C. Prommel, and Mr. Paul M. Hopkins, Registered Mining Geologists and Engineers, to set forth from their experience certain factual guidelines which contribute to the success of a placer mining venture.

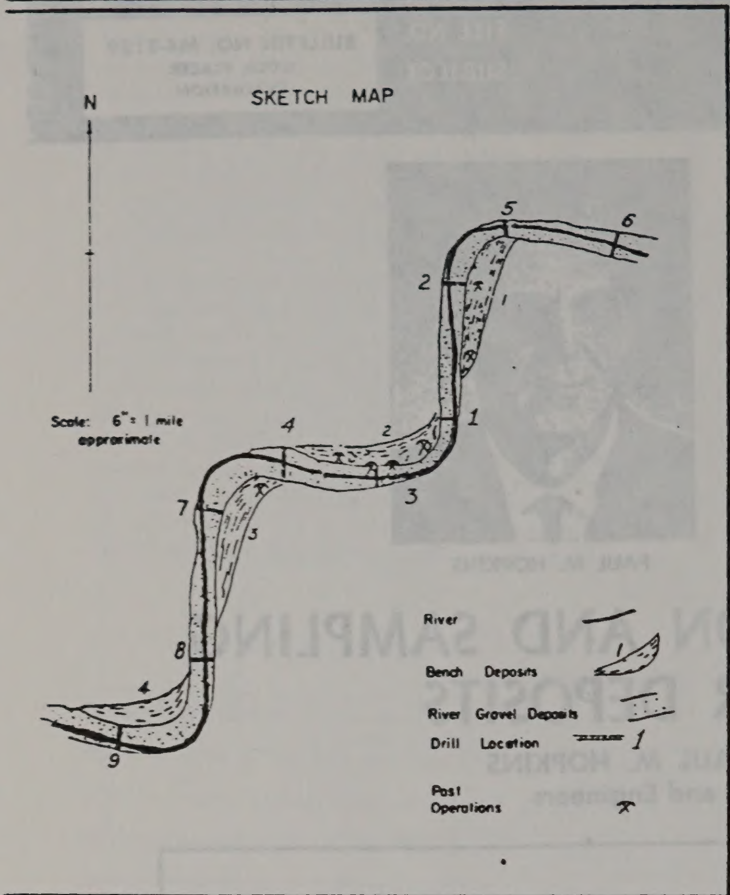
Questions concerning the evaluation of placer areas should be directed to Mr. Prommel at his office, 731 South Downing St., Denver, Colorado 80209, phone area code 303-733-8445; or Mr. Hopkins, P. O. Box 403, Golden, Colorado 80401, phone area code 303-279-2313.

Discovery of Placer Deposits

The experience, skill, and enthusiasm of the ones who discover a gold-bearing area are frequently discounted by those to whom they turn for financial aid in developing a commercial venture from their discovery. Principals with sufficient financial resources to back such a venture normally do not possess the experience or technical ability to exploit the discovery and must turn to impartial professionals for reliable verification of the discovery and an evaluation of the cost and potential profit from participating in the placer mining venture. Fortunately most financial people appreciate the importance and wisdom of reliable advice before investing large sums of money.

Preliminary Reconnaissance

An essential step in the evaluation of a potential property is a preliminary reconnaissance trip and visual examination of the area prior to a complete placer testing program. If the results of the reconnaissance are favorable a placer testing program can be formulated



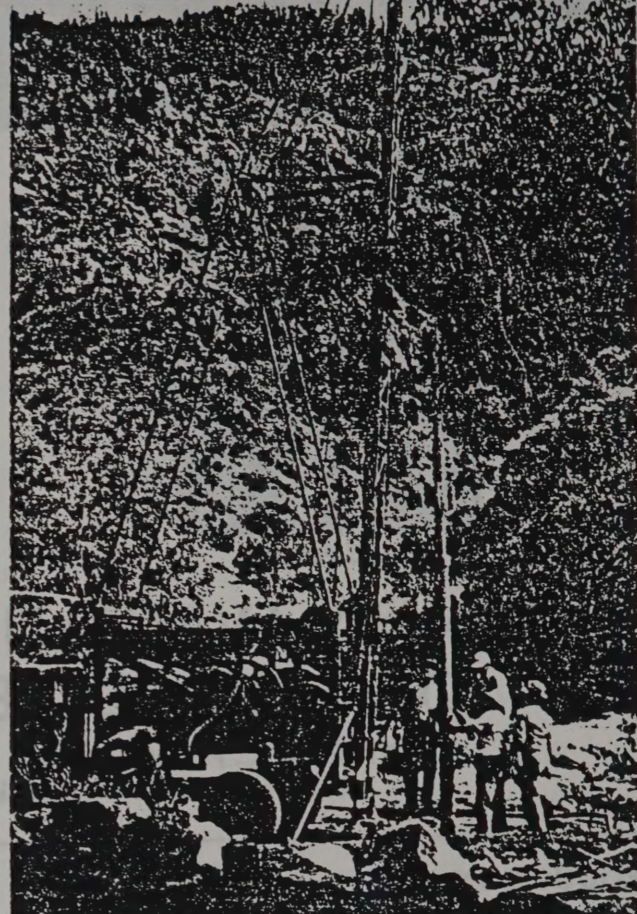
A typical sketch map prepared on preliminary reconnaissance of a placer area should clearly illustrate the location being studied. Numerical keys are referred to in the report prepared by the consulting engineer.

and rather precise cost estimates can be prepared for conducting the work necessary to provide facts on which future decisions can be made.

Objectives of the preliminary reconnaissance trip and visual examination should include:

(1) Examination of the title to the property, including the royalties and other lease terms applicable to the land.

Mr. H. W. C. Prommel, center, is shown with a surveying crew supervising the mapping of a potential placer property. An accurate map is essential to plotting the sample test hole locations.



A churn drill placer sample hole being driven in Clear Creek, Jefferson County, Colorado. Supervision of drilling is very important to reliable results.

(2) Prove the presence of mineralization by visual examination, geological data, limited excavations, hand panning, and research into known placer developments in the immediate area.

(3) Confirm the approximate depth of bedrock and benches.

(4) Deliniate the most favorable areas for exploratory work.

(5) Determine the physical limitations as imposed by local conditions on the movement of equipment, supplies, and personnel during the period of subsequent explorations.

(6) Predict from the known and visible factors if the property warrants detailed exploration. Upon finding favorable values in the gravel and an apparently attractive total cubic yardage available for economic operations the engineer can then and only then recommend a program to fully implement a complete placer testing program of the property as the second phase of the placer evaluation and examination. This second phase will determine the gross values in dollars and cents and establish the data upon which to base the economics of the project for the investing group.

Developing a Placer Area Test Program

A fully implemented placer testing program has the following objectives:

This is part of a placer testing crew starting downhill after a blizzard. This operation took place many years ago in Park County Colorado.

(1) Churn drilling and/or other testing of the full section from surface to bedrock at numerous locations to obtain measured samples. Sampling should not be limited to one or two locations but should follow a predetermined pattern of sample hole locations. "Keystone Drilling" is among these techniques and is described later in this paper.

(2) Calculate the value of gravel in place by determining the value contained per cubic yard of "pay dirt" from each drill hole or pit.

(3) Determine the depth and contours of bedrock beneath the stream and also in the adjacent areas of the property.

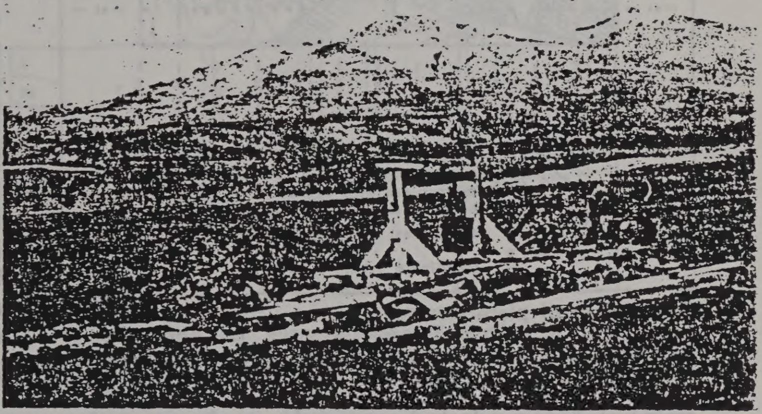
(4) Compilation of information gained from sample hole yardage will indicate total yardage available and total gross values available. Other important considerations and factors established during the test program which constitute the engineering report and influence final decisions on both domestic and foreign properties are as follows:

(a) Distance from main highway and shipping points.
(b) Political climate, taxes and attitude towards mining.
(c) Legal status of claims.
(d) Access to electric power, fuel, telephone, etc.
(e) Elevations, climate and rainfall.
(f) Working season.
(g) Availability of adequate process water or excesses.
(h) Potability of water.
(i) Type of commercial operation suitable.
(j) Amount of royalty either required or possible.
(k) Character of deposit, is land clearing necessary.
(l) Topography, physiography and geology of area.
(m) Depth from surface to permanent water table.
(n) Gradation and character of gravels.
(o) Presence of clays and cementing materials.
(p) Character of values, visual and chemical analyses, including fineness of gold.
(r) Anticipated problems in recovery.
(s) Location of values in the deposit.
(t) Extent of other placer operations in vicinity.
(u) Danger of stream pollution and means of control.
(v) Housing facilities, school conditions, and distance from property.
(w) Cost and availability of local labor.
(y) Documentary photos, maps, logs and reports.
(z) Market for contained minerals other than gold.

With the above facts available a projection of the net value of the deposit can be made. This includes

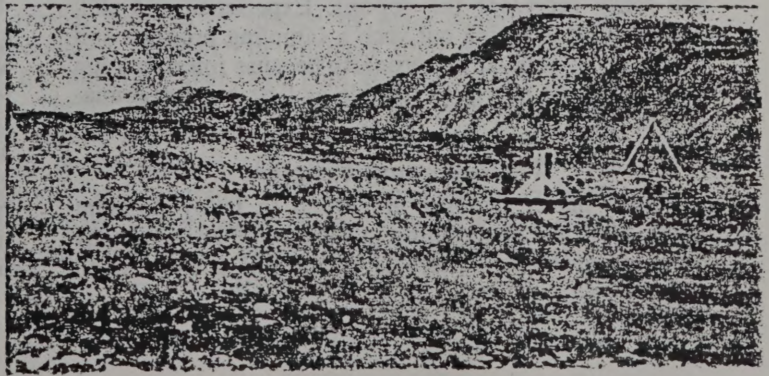
Four sample test pits are shown in this photo. It is only by excavating material to bedrock and processing the material from each pit using standard placer recovery methods that reliable projections can be made to determine economic success of a placer deposit.

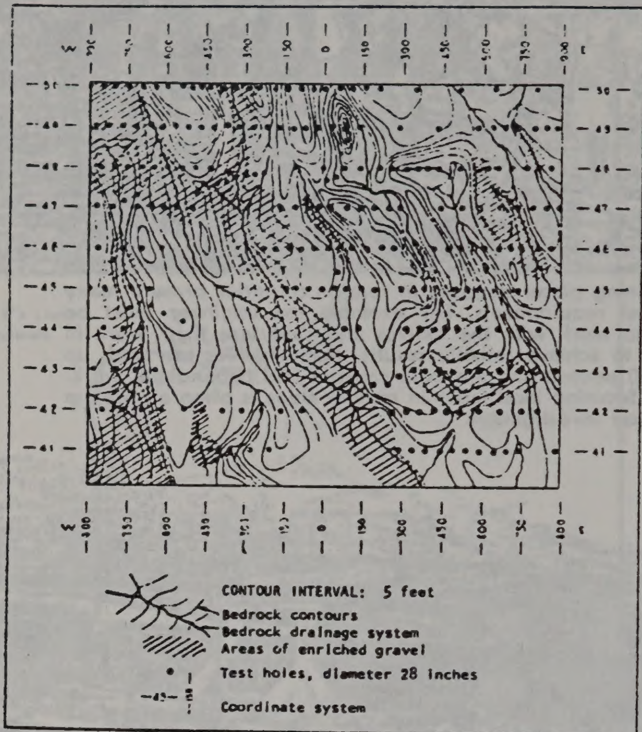
Examining placer deposits in foreign countries present many unusual requirements. For example the engineer must be able to work with the natives and supervise their work in order to achieve reliable results. Many natives can pack up to 100 pounds and are both capable and dependable workmen. Securing local labor in remote areas is often a problem in placer investigations.



Proper excavation of a test pit to bedrock is an exacting procedure. Walls must be shored to prevent caving. All material removed must be measured, bagged, and identified for future processing.

Conditions are not always ideal for placer investigation work. This pit, three feet by five feet in size went to a depth of 42 feet. Large boulders at left were typical of material that had to be removed in this placer investigation.





Typical map of placer area prepared by consulting engineer experienced in evaluated placer deposits. The entire area is mapped and a grid is established that locates the necessary drill holes and pits. Experience and skill are required in developing a placer testing program. Many valuable placer areas have been overlooked or rejected by inadequate exploration. Also, inadequate exploration has often resulted in increased risk in placer operations.

determination of equipment required to handle a designated capacity operation sufficient to yield a profit based on available values.

Mapping the Placer Area

One of the objectives of the reconnaissance is to secure a reliable map of the placer area. Topography must be known and access to the area must be checked out. Often it is necessary to use mule-back transportation or air transport to bring equipment to the property. Modern transportation has made available to the industry many areas which were previously considered inaccessible or impractical.

The map of the area is essential to establish the extent of the property and the deliniation of the total yardage available for processing. In order to fully understand the distribution of values throughout the property it is necessary to have a sampling program that will fairly represent the deposit. Experience and skill are required in laying out such a sampling program and this must be done at the property because interpretation of maps is not nearly as reliable as first hand examination of the area.

The Sampling Program

Since placer deposits are found in many different areas, they take many different forms, and present many different problems. There is no simple answer to all problems in all deposits. Each deposit is an individual entity and must be evaluated on the basis of its own personality. The scope of this paper is necessarily limited but an attempt has been made to encompass many different types of situations to aid those faced with problems similar to the ones presented in this discussion.

Two types of placer ground can be generally described as "river gravels" and "bench gravels." The river gravels are normally under the flood plain and involve hydraulics while the bench gravels are essentially dry and water must be available for normal economic processing.

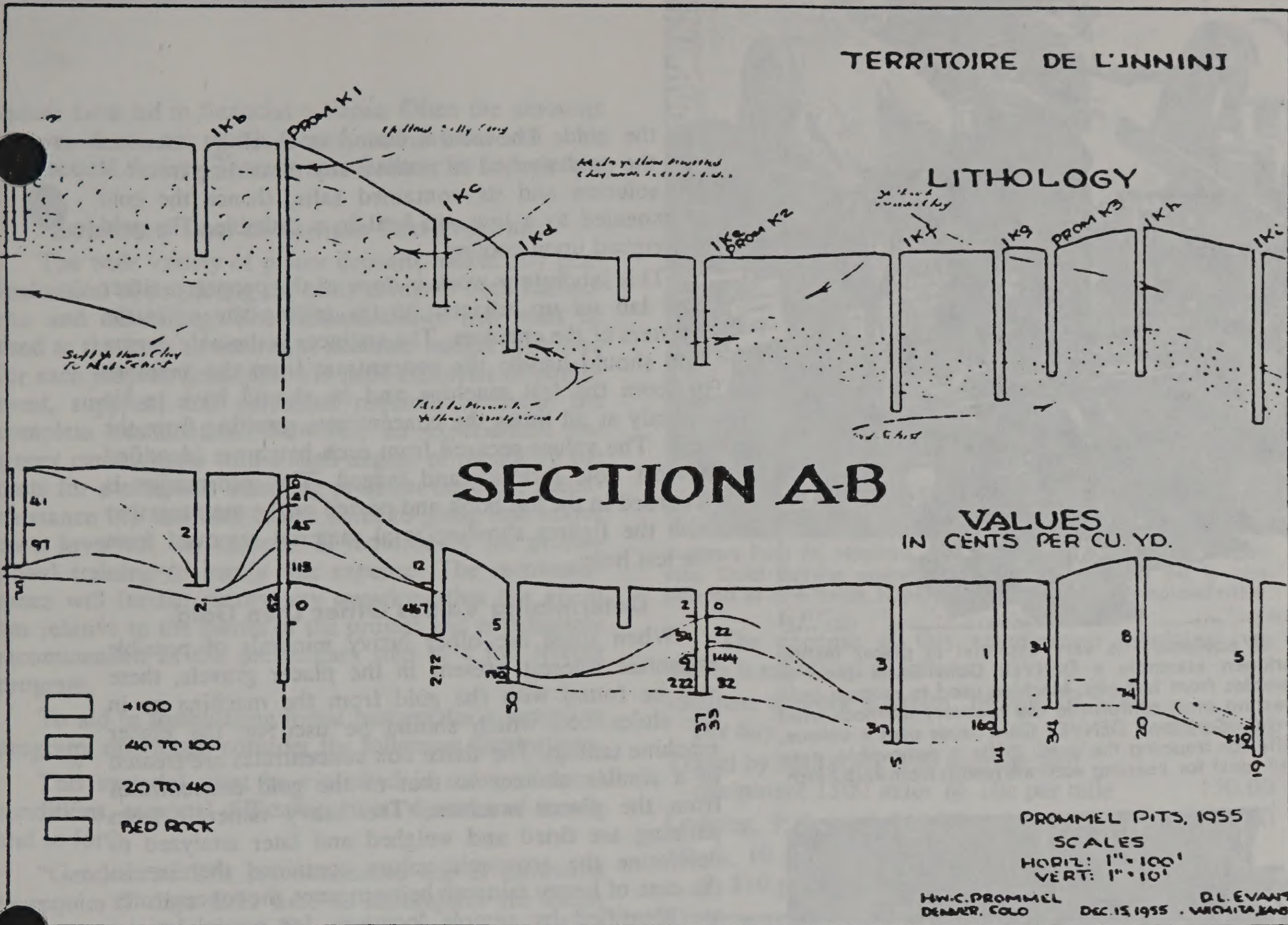
Sampling Bench Gravels

The "bench gravels" are examined by test pits or other such methods as required to obtain a finite measured cubage per foot of depth of gravel in place and thence recovering the gold by placer mining methods from the sample of known volume. Panning with a hand gold pan, sluicing or, more practically, the use of test machines of known reliability can be used to determine these values. The depth of the gravel from the surface to the bedrock is determined at the several test locations. The area of gravels containing values of interest are deliniated on the map of the area, and by obtaining the value per cubic yard at the several test locations, the total yardage of the deposit is indicated to have certain finite values in gross recoverable placer gold.

Sampling River Gravels

In contrast, the "river gravels" can only be tested by the technique of mechanically driving a heavy pipe from surface to bedrock, removing the gravels from within the pipe as it is driven downward and thence recovering the contained gold in the gravels so removed from within the pipe. The technique is known as "Keystone Placer Drilling." The drills can be purchased outright or the services contracted for in certain areas. The program should be under the supervision of the examining engineer at all times. This is the most reliable method of determining values in a water saturated gravel as it has proven its reliability on all types of placer resources throughout the world.

A number of holes using the pipe are driven at right angles to the general flow direction of the stream. It is also usual to make not one but several lines of these holes on a property at distances of from 1000 to 3000 feet apart, depending upon the assumed substructure of the deposit. There is no set rule as to the number of holes to drill. Ground showing an erratic distribution of gold naturally requires more holes than a property showing uniform values. The drive pipe is sized and by knowing the full depth to bedrock, one determines the gold values per cubic yard of gravel in place for the full depth of the deposit. The calculations are quite simple and have proven to be reliable. However, one cannot overlook the factor of experience in both locating the holes and in driving the pipes for an inexperienced worker without adequate supervision can unknowingly provide misleading results.



s is a typical cross-section of drill holes made on a placer testing assignment. Information from the test pits is plotted on the maps and is used by the engineer in reaching conclusions not only as to the commercial success of the proposed venture but also the areas of primary and secondary interest. Only by very careful and deliberate testing can facts be secured on which to project reliable recommendations.

Composition of the Sample

The professional examination of a placer deposit involves much more than drilling holes. A placer examination is accomplished by using proven placer recovery methods on the gravel from the holes drilled, pits dug, or otherwise sampled. The size distribution of material from coarse to fine, for example, the rock size, the pebble size, the sand size, and any clay is determined as part of this placer gold recovery testing program. The distribution of these various sizes is determined by measuring the several physical "parts of the gravels" after the process of separation and washing.

Determining Sample Hole Locations

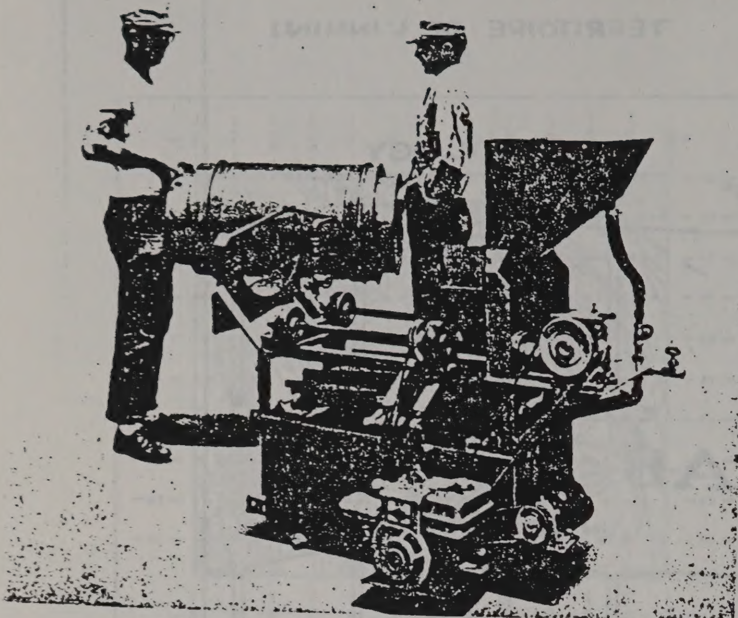
The locations, spacing, and the sampling grid pattern which is established at the time of the preliminary reconnaissance is plotted on the map. In very large placer areas a sampling program may be developed in two or more stages, subsequent stages depending upon evaluation of the initial stage. It is only by an extensive sampling program that boundaries of the commercial operation can be established with any degree of intelligence or commercial significance.

The Sample Hole

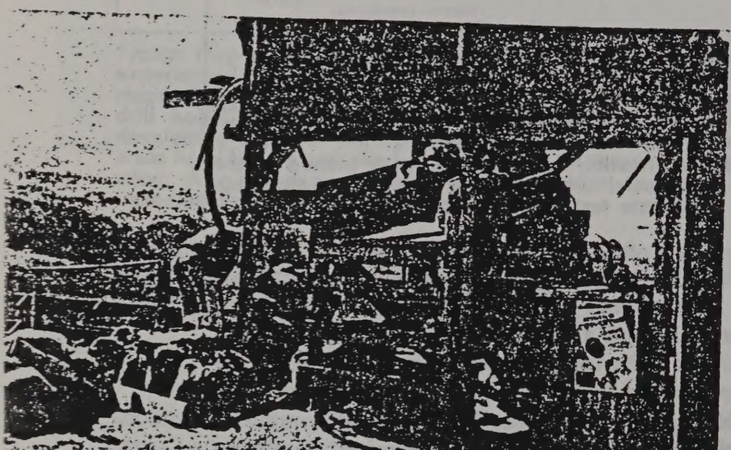
The sample hole is located on the basis of the grid projected on the placer area map. Great care and skill must be used in excavating a sample hole. All material excavated is saved and measured by volume and placed in containers, tagged at the location to positively identify the sample. The total sample from each hole or pit is considered as one batch. Systematic records of each hole should be kept in a log book. The total volume or batch is then processed in the field using a reliable placer testing machine to remove oversize material, disintegrate the clays, and recover the gold-bearing heavy sands. As in any mineral beneficiation project the efficiency of the machine used to process the sample material must not be overlooked. Since the data secured from these sample holes is in effect a very small sample any errors present will be magnified hundreds of times to predict the quality of the entire deposit.

Recovering Gold From the Sample

The concentrate recovered from the placer machine is further reduced in volume by hand panning by an experienced "panner" or engineer until the minimum of the material other than gold is present. Fire assays must never be used to determine the gold content as the results are completely unreliable and against all good placer practice. The gold is amalgamated with mercury and separated from the other heavy minerals in the pan. The amalgam is next treated with a dilute solution of nitric acid which dissolves the mercury, leav-

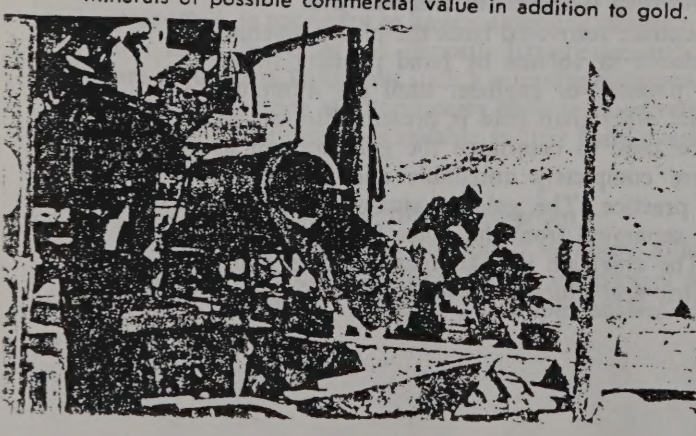


Portability of equipment is very essential in placer testing work. Workmen assemble a DENVER Gold Saver used in treating samples from test pits. Machine used to recover gold in placer testing must conform to the recovery methods used in commercial operations. DENVER Gold Saver uses a unique, molded riffle for trapping the gold. Riffle is removable, easy to clean and ideal for keeping accurate results from each batch.



Material from sample test pits has been bagged, identified, and delivered to the Gold Saver machine. At right. Sample receiving platform aids the process of feeding the Gold Saving machine.

Oversize material from scrubber and trommel screen is wheeled to the dump. Fines that discharge from the shaking molded riffle pass through sluice box for black sand recovery and determinations of black sand content per cubic yard. Black sands are examined for presence of other heavy minerals of possible commercial value in addition to gold.



ing the gold. The acid solution is carefully decanted and the gold washed to remove any remaining traces of the solution and its contained salts, thence the gold is annealed to a low red heat in a crucible. The gold is weighed upon cooling.

This laboratory work is done at the property, either at a lab set up adjacent to the testing site or in the quarters of the engineer. The engineer is the only person who should handle the concentrate from the pannings or from the test machine and he should have in his custody at all times the concentrates resulting from the tests. The values secured from each batch are identified by test hole location and tagged. The information is recorded in the log book and posted on the map together with the figures showing total material removed from the test hole.

Determining Values Other Than Gold

When there are other heavy minerals of possible economic interest present in the placer gravels, these will be found with the gold from the machine or in the sluice box which should be used on the placer machine tailings. The sluice box concentrates are treated in a similar manner to that of the gold concentrates from the placer machine. The heavy minerals from panning are dried and weighed and later analyzed to determine the economic values contained therein. In the case of heavy minerals being present, the concentrates are identified by sample locations for consolidations with the report. The presence of valuable heavy minerals can augment the potential of the property and thus should not be overlooked in the examination.

Supervising the Field Work

The sample testing program is an exacting and deliberate procedure. The engineer may be able to make a continuing evaluation of the property as the work progresses and there is frequent temptation to discontinue a sampling program once sufficient values have been determined. However, most placer deposits are spotty and decisions based on incomplete information have frequently been disastrous. Far more is to be gained by following the program to its completion than by impatience in putting the property into commercial operation before full facts are known.

When the field work is completed the engineer will be able to make his recommendations based on the results of the total testing program. It must be emphasized that the techniques of separating the gold from the gravels and the recovery of that gold from the holes or test pits must conform to the methods used on dredges or other gold placer recovery operations. Many theoretical and impractical devices can be employed to yield results which cannot be duplicated in commercial operation. Because of the high ratio of concentration required and the low yield per cubic yard most placer operations must be scaled to a sufficient capacity that will permit profitable operation. A process must be developed that will provide economic recovery. Frequently elaborate methods of saving hard-to-recover

values have led to financial collapse. Often the amounts recovered are not worth the expense involved in such impractical devices. There is no substitute for experience.

Estimated Cost of Sample Test Program

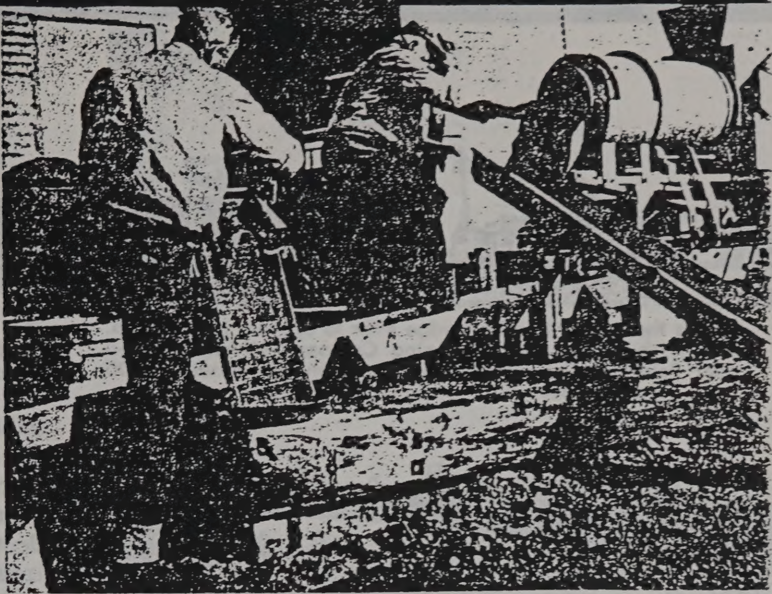
The wide variety of placer deposits makes any generalization of predicting the costs involved both hazardous and misleading. The reconnaissance trip must be used as the basis to determine accurate budget estimates for each project. Principals will need estimates of equipment, supplies, and personnel required to make the complete examination. However, an experienced engineer can estimate with a high degree of reliability the costs for a complete sampling program once the reconnaissance trip has been made. Prior to going into major costs involved, the engineer is required by his professional training to justify this expense. The reconnaissance will further clarify any questions that the client has relative to the merits of the project and is definitely recommended as the preliminary step to a full testing program.

To aid in formulating initial budgets for a sampling program, one might consider the following illustrations.

This material must be evaluated in terms of local conditions, seasonal difficulties, total distance of travel, and so forth.

"Gentlemen: The reconnaissance trip is estimated to require 7 days field time to accomplish the items listed as the objectives. Three days are estimated as required for travel, one and one-half days each way between the consultant's office and the property. An additional four days will be required to evaluate the results and check any preliminary statements, verify field reports and to submit the recommendations for further actions.

H. W. C. Prommel uses gold pan to reduce the volume of heavy minerals prior to amalgamation. Tub is used to save other heavy materials for subsequent evaluation.



Authors Paul M. Hopkins, left, and H. W. C. Prommel supervise Gold Saving operations. Each sample test pit is considered as one batch. Hopkins washes gold from molded riffle.

"The expense of this examination involving two weeks is estimated as follows:

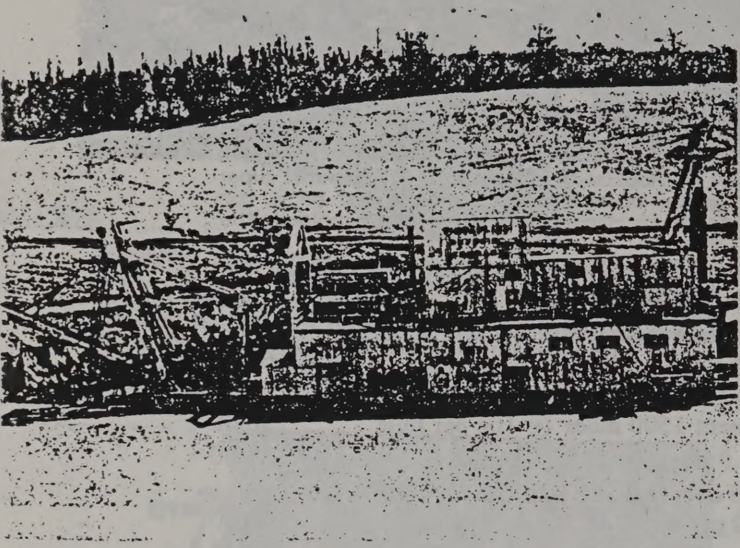
Engineer and one assistant, 14 days @ $150 per day	$2100.00
Travel by field truck with supplies and equipment 1500 miles @ 10c per mile	150.00
Lodging, 9 days @ $15.00 per day	135.00
Meals, 10 days including the travel period @ $10 per day	100.00
Contingencies	125.00
Total estimated cost of reconnaissance trip and report	$2610.00"

What the Report Includes

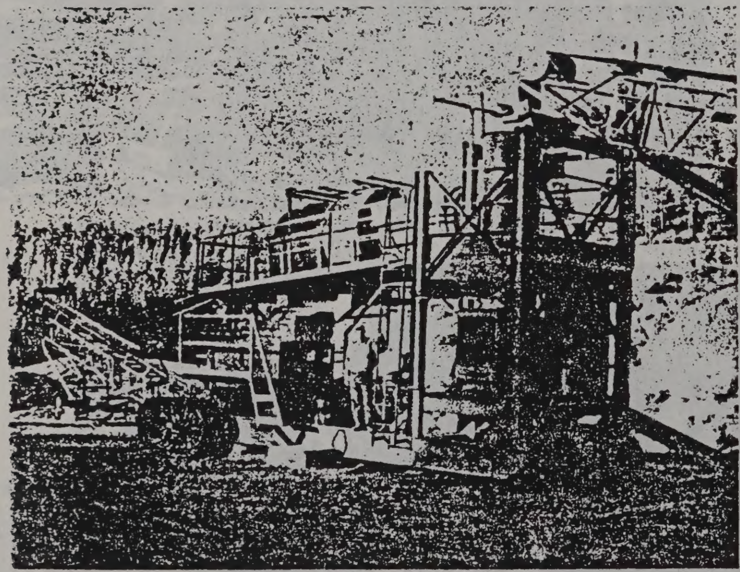
As in any professional situation the facts revealed by the investigation are only a means to a conclusion. It is the experience required to interpret the facts that frequently makes the difference between success and

Paul M. Hopkins pans material washed from molded riffle. Examination of the concentrate recovered by the Gold Saver reveals both size, and character of the gold prior to amalgamation.

This is the largest bucket dredge ever to operate in Colorado. This is the South Park Dredging Company which operated near Fairplay, Colorado, with a capacity of about 15,000 cubic yards per day.



The Timberline Dredging Company's bucket dredge operated on Beaver Creek in Park County, Colorado in 1942. Floating dredges present both problems and advantages that must be evaluated for each deposit.



This dry land placer machine operated in California in 1963. Placer ground was fed to the track-mounted placer recovery unit by conveyor which received the gravel from a drag-line. This unit included a scrubbing trommel, a DENVER Mineral Jig and a DENVER Table as the gold recovery devices.

This is a Kister Bucket Dredge operated in 1946 near Oroville, California. Placer reports should include equipment necessary for commercial operation.

failure. A competent engineering report from a preliminary placer investigation will include such items as reports on title or royalties involved in operating the claims, information on geology of the area, reports on depth of overburden and depth of gravel beds, itemizations of local conditions which will affect favorably or adversely the operation, a detailed outline of the sampling steps recommended, a map showing the area examined, and detailed cost estimates of the further work recommended prior to commitment of major expenses.

Verification of Historical Data

One of the frequently important and often misleading elements of any placer report is that of the local history surrounding a deposit. Because an area was once worked is no more an assurance that values have been exhausted than it is that future operation will be profitable. Listening, snooping, asking, sifting, researching, questioning, and observing must be done with an experienced skill so that verification of fact as well as recognition of fiction can be used intelligently in the evaluation. Hearsay reports of profits of previous operations must be checked out for reliability. Accuracy of previous engineering reports must be questioned and if possible the equations and logs must be reworked to discover possible typographical errors before the previous reports can be accepted. As much as one may wish to accept unverified data, careful review and examination by competent professionals is worthwhile.

Other Factors Influencing Success or Failure

A review of the major failures of many placer ventures reveals a failure on the part of principals to take advantage of sound engineering experience preliminary to the obligation of major investments. At $35.00 per ounce mandatory sales price in the U.S.A., each gold mining operation is of necessity a marginal operation. There are strong indications, particularly in areas outside the United States that relief either in the form of a higher price for gold or incentives such as subsidies, bonuses, and tax benefits are a future probability rather than simple speculations. As gold bearing placer deposits are exploited the difference between success and failure depend on the use of reliable enginering information to establish the soundness of an operation both as to size of the operation, capital required, and the extent of possible rewards.

Printed in U.S.A.

GEOLOGICAL SURVEY
BULLETIN 280

THE GEOCHEMISTRY OF GOLD AND ITS DEPOSITS (together with a chapter on geochemical prospecting for the element)

R.W. Boyle

be made up of nearly massive aggregates of uraninite. There is commonly a higher content of uraninite in pyrite-rich parts of ie beds. Thucholite was not seen in the matrix of the conglomerate or quartzite by the writer; it occurs, however, as warty aggregates along faults and fractures in places and as thin columnar seams with uraninite and pyrrhotite along bedding planes. Ruzicka and Steacy (1976) describe some of the latter occurrences in detail.

Gold and silver occur in the Blind River–Elliot Lake ores only in small amounts, the average gold content for most samples being 0.09 ppm (0.003 oz Au/ton). Locally, high assays can be obtained particularly where sulphides such as galena, sphalerite, cobaltite, etc. occur. Most of the gold and silver are in the pyrite and sulphide concentrates as indicated by assays of these materials in our laboratories. The average for the pyrite and mixed sulphide concentrates is about 0.64 ppm (0.019 oz Au/ton). No native gold was seen in these concentrates by the writer, but the metal has been reported as occurring sporadically in some of the ores.

The oreshoots in the Blind River–Elliot Lake mines are usually gently dipping tabular bodies composed of lenses and interfingering beds of pyritic quartz-pebble conglomerates and quartzites as shown in Figure 66. In most of the stopes the pyritic mineralization of both the conglomerates and quartzites commonly shows a wispy to banded appearance, and there is undoubted evidence of replacement of crossbedded features by pyrite and uraninite. One can also see seams of pyrite along bedding planes, veinlets of pyrite in the quartz pebbles in places and little trains or veinlets of uraninite (pitchblende) in the matrix of the conglomerates. All conglomerates and quartzites rich in uranium are generally always relatively rich in pyrite, but the converse is not necessarily true. Sericite-rich areas seem to be indicative of high uranium contents in most samples. Finally, in some areas quartz veining is evident. These and isolated fractures contain pyrite, chalcopyrite, pyrrhotite, galena, sphalerite and occasionally cobaltite. They are not radioactive.

On the question of origin the two general theories – hydrothermal and modified placer – common to these types of deposits prevail. The early workers (Joubin and James, 1957; Holmes, 1957) supported the modified placer theory and later workers have tended to agree (Roscoe and Steacy, 1958; Robertson and Steenland, 1960; Bain, 1960; Pienaar, 1963; Robertson, 1974). Davidson (1957, 1964–1965), however, maintains that the uraniferous ores are hydrothermal or due to downward leaching by saline waters as explained in the section on the Rand.

Other Precambrian occurrences of uraniferous and auriferous quartz-pebble conglomerates in Canada

Quartz-pebble conglomerates are developed in a number of other districts in the Canadian Shield; many contain uranium and thorium with minor amounts of gold, others are considerably enriched in gold as well as in uranium and thorium. East of the Blind River–Elliot Lake district, some 35 mi west of Sudbury, the basal Huronian rocks yield uranium from the Agnew Lake Mine. The pyritic oligomictic conglomerates comprising the ore horizons are steeply dipping and extend to depths of 3000 ft or more. The principal ore mineral is uranothorite, and the Th/U ratio in the ores is about 3:1 compared to 1:2 for the Elliot Lake deposits. Gold is found only in traces. Somewhat similar pyritic conglomerates and quartzites occur elsewhere along the base of the Huronian Supergroup west and east of Sudbury, one particular occurrence of note being present at the southwest corner of Lake Wanapitei. This conglomerate contains essentially monazite and leucoxene as the radioactive minerals; gold occurs only in traces.

Uraniferous occurrences in the Montgomery Lake–Padlei area of Northwest Territories are in Aphebian pyritic oligomictic conglomerates in the Montgomery Lake Group, which underlies the predominantly clastic Hurwitz Group and rests on rocks of Archean age. The occurrences contain abundant pyrite with uranothorite, zircon, leucoxene and notable amounts of gold (up to 0.1 oz Au/ton).

Occurrences of highly deformed pyritic uraniferous conglomerate, probably of Aphebian age, and belonging to the Sakami Formation, lie on Archean greenstones and greywackes on the west side of Sakami Lake in northern Quebec. Locally they carry in excess of 10 per cent pyrite and contain disseminated uraninite (with 10% ThO_2) and a thorium silicate, either thorite or allanite (Robertson, 1974). Gold occurs only in traces as far as is now known. Other basal quartz-pebble conglomerate lenses occur in quartzites and arkoses in the Papaskwasati Basin north of Lake Mistassini in Quebec. These belong to the Papaskwasati Formation of the Mistassini Group of Aphebian age, contain minor pyrite, are radioactive and have received some attention. Their gold content as far as is now known is low.

> The large nuggets found in the drift are simply the reliquiae of the chief masses of gold which once occupied the uppermost parts of the reefs, and that like the blocks of many an ancient conglomerate, they have been swept from the hilltops into adjacent valleys by former great rushes of water.
>
> –Murchison, Siluria, 5th ed., 1872

Placer deposits

Placer deposits provided early man with the first samples of gold and since that time have accounted for a large production of the metal. If we include the Witwatersrand and other quartz-pebble conglomerates as fossil placers or modified fossil placers, then the placer type of deposit has provided more than two-thirds of man's store of gold.

Before proceeding further certain terms with respect to placers should be defined.

The term 'placer' is evidently of Spanish derivation and was used by the early Spanish miners in both North and South America as a name for gold deposits found in the sands and gravels of streams. Originally, it seems to have meant 'sand bank' or 'a place in a stream where gold was deposited'. While many other terms have been coined for deposits in weathered residuum and alluvium none is quite as succinct and expressive as 'placer'.

The terminology of the zone or stratum containing an economic concentration of gold in eluvial and alluvial placers is varied. We shall use the miner's term 'pay streak', which is commonly used in Canada and the United States. Other

English terms in use include 'pay gravel', 'pay sand', 'pay dirt', 'pay wash', 'pay channel', 'pay lead', 'run of gold', 'gutter' and 'wash dirt'. There are a host of Spanish terms used mainly in Mexico and South America, and there are also miner's terms for the rich gold zones in practically all languages where eluvial and alluvial placers are worked. Rickard, many years ago, advocated the use of "gold-bearing channel" for all these terms, but it has not been generally accepted.

The tenor of pay streaks or of placer gold gravels and sands, in general, is referred to by the value (in ounces, pennyweights, or in any unit of currency) per cubic yard, per running foot of channel or per square foot of surface; also occasionally in bonanzas by dollars or some other unit of currency per pan. It is of interest to note that placer deposits can be worked whose gold content is as low as 0.1 ppm.

The pay streaks of placer deposits may rest on or near bedrock or on some stratum above bedrock. The bedrock in placer deposits is commonly referred to as the 'true bottom', although the term is little used today. When the streaks rest on a well-defined stratum of sand, gravel, shingle or clay above the bedrock they are said to be on a 'false bottom'.

Placers have been variously categorized, but here we shall use a simple nomenclature based upon whether the placers are formed by concentration of gold in situ over or in the immediate vicinity of primary deposits, namely 'residual' or 'eluvial placers', or by agencies that have concentrated the gold in the near vicinity or at some distance from the primary source. In the latter category we recognize 'alluvial', 'beach' and 'aeolian placers'. The terms 'saprolite' or 'saprolitic placer' were formerly used for certain types of eluvial placers, mainly in the eastern United States.

Eluvial, alluvial, beach and aeolian placers may become buried after their formation and are sometimes referred to as 'buried placers'. These placers may be buried under (1) volcanic deposits as in California and Australia; (2) glacial deposits as in Canada and U.S.S.R.; (3) talus and other slope deposits; (4) aeolian deposits as in Australia; (5) alluvial sands and gravels; and (6) marine and lacustrine deposits.

There are fossil (lithified) equivalents of eluvial, alluvial, aeolian and beach placers. These are described subsequently.

The gold in auriferous placers may come from one or more of the following sources:

1. Auriferous quartz veins and other types of gold-bearing deposits mentioned in the section on epigenetic deposits.

2. Auriferous sulphide impregnation zones, porphyry copper deposits, etc.

3. Auriferous polymetallic deposits.

4. Slightly auriferous quartz stringers, blows and veins in schists, gneisses and various other rocks.

5. Various slightly auriferous minerals such as pyrite and other sulphides in graphitic schists and other rocks.

6. Slightly auriferous conglomerates, quartzites and other rocks.

7. Old placers.

It should be noted that the geological history of productive placers is frequently complex, much more so than the sequence: primary vein or lode source → eluvial placer → alluvial placer. Often an intermediate collector of gold is involved, mainly auriferous conglomerates, quartzites, etc. Fayzullin (1968) emphasizes this point in his papers. He recognizes a number of variants in the lode-placer sequence as follows: (a) lode → deluvial placer → interceptor → alluvial placer, (b) lode → interceptor → alluvial placer, and (c) interceptor → alluvial placer.

The primary agency that produces gold placers is weathering; a process that involves numerous complex chemical reactions that are explained in detail in Chapter IV. Three things may happen to gold in primary deposits: (1) the gangue minerals may be disintegrated and leached away, leaving the gold relatively untouched; the gold may remain in situ in the oxidized zones or pass into eluvial and alluvial placers; (2) the gold may be dissolved and carried far away from the deposits in which case no placers are formed; or (3) the dissolved gold may be wholly or partly reprecipitated on nuclei of gold in the residuum or on similar nuclei as they are moved along in the alluvium of streams, rivers, beaches, etc. The last process is largely responsible for nuggets. The formation of placers is, therefore, a combination of both mechanical and chemical processes working in consort over long periods of time. We shall return to this theme later in the present section.

The mechanical agencies that assist in the transport and winnowing of gold into placers are gravity, the running water of streams and rivers, the agitation of waves along the shores of lakes, seas and oceans, the wind, and glaciers. Gravity is a force that operates in the formation of all types of placers; it is the principal agent that concentrates gold in eluvial placers. The other agents give rise respectively to the following types of placers: stream and river (alluvial) placers, deltaic placers, beach placers and aeolian placers. There are few authenticated aeolian gold placers of any size, and we shall not consider them further except to note that near the outcrops of some of the primary deposits in Australia the wind had blown away the finer detritus exposing coarser material in which gold was enriched in places (Hoover, 1899, Rickard, 1899). Auriferous sand dunes have been recorded by Spurr (1906) in the Silver Peak quadrangle, Nevada. These apparently contain only traces of gold and silver. Placers due entirely to glaciation are, likewise, uncommon, and none of economic value are known to this writer in moraines, kames or eskers. Certain outwash plains, where worked by glacial and postglacial streams may, however, contain low grade placers in some glacial terrains. Winnowing of gold by present-day streams and rivers from glacial sands and till and working of these materials along certain lakes has also led to minor concentrations of gold in many places in Canada and other glaciated countries. All of these types of glacial deposits are mentioned briefly in the alluvial category.

The minerals concentrated in placers include two types – those with low to medium specific gravity (the light minerals) and those with a medium to high specific gravity (the heavy minerals). All of these minerals have three features in common – great physical resistance to mechanical abrasion and comminution, great chemical resistance to solution in surficial waters and a general equidimensional character in their form. The last is of importance since flaky minerals such as molybdenite, scaly gold and specularite are difficult to concentrate in spite of their high specific gravity. Another factor is the degree to which a mineral can exist in a subdivided form and yet remain relatively chemically stable. This is a factor with respect to gold since the metal can exist in almost infinitely

subdivided form giving the so-called 'flour', 'float', 'flood' or 'skim' gold. Such gold does not sink readily in sand and gravel and hence is not concentrated to any extent in placers; on the contrary it may be transported hundreds of miles in running water to be deposited over broad deltas or on the floors of lakes and oceans where it is commonly disseminated throughout great thicknesses of sediment. This particular feature is of importance in discussing the origin of the very finely divided gold of deposits such as the Witwatersrand. (*See* the final section of this chapter on the origin of deposits.) Commonly much flour gold accumulates temporarily in streams and rivers on bars on the inside of curves or meanders or in other areas of slack water ('skim bars'). Unless these are recognized they may give an inflated impression of the potential of a placer stream or river. Skim bars are also misleading during geochemical prospecting within a restricted area since they give false anomalies. This aspect is discussed further in Chapter V.

'Moss gold' is flour gold trapped by mosses and other plants along river and stream beds during high water. 'Moss miners' have collected this gold for centuries along the auriferous rivers throughout the world. It is easily recovered by burning the moss.

With skim and moss gold a new 'crop' appears each year after high water in auriferous water courses. This is partly the reason for the adage that 'gold grows in placers'. It is, however, probably not the whole story as will be seen from a later discussion.

The most common minerals with low to medium specific gravity in placers are quartz (S.G.=2.65), muscovite, amphibole, pyroxenes, tourmaline, garnet, diamond, chromite, rutile, barite, corundum, wad, limonite and zircon (S.G.=4.5); those with medium to high specific gravities in placers include monazite (S.G.=5), magnetite, ilmenite, cassiterite, wolframite, scheelite, cinnabar, gold and platinum (S.G.=22 when pure). A great variety of sulphides and sulphosalts may accompany gold in placers. This writer has seen pyrite (often abundant), galena, sphalerite, arsenopyrite, boulangerite and jamesonite in placers in Yukon Territory and elsewhere. Native bismuth is present in some placers; native mercury in others; and more rarely native arsenic, native silver, arquerite (Ag-Hg amalgam), native copper, native lead (not shot), native zinc, cinnabar, realgar, sperrylite, molybdenite, chalcopyrite, hematite, carbonates, feldspars, kyanite, topaz, spinel, allanite, epidote, sphene, tantalite-columbite and apatite. Some gold placers contain diamonds and other gems such as rubies, sapphires, emeralds, topaz, garnets, etc. Commonly the gems are secondary in value to gold, and in some placers they are so sporadic as to be a curiosity. Certain placer sands and gravels are cemented or coated by limonite, wad or mixtures of these mineral aggregates. These aggregates commonly contain much silica, alumina, titania and other hydrolysate oxides and frequently large amounts of humic material. Placer gravels or sands strongly cemented by limonite and other iron oxides are often referred to as 'cement gravels' in English speaking countries or 'cangalli' in South American countries. A few placer sands and gravels have been cemented by carbonates, silica or clay, and most placers in the permafrost of Canada, Alaska and U.S.S.R. are frozen solid, requiring a complicated procedure to thaw them out in places.

The gold of placers has varied characteristics. The most common habit is as dust that comprises particles ranging from specks the size of the tip of a needle or less to those the size of a flaxseed (in quantitative terms, <0.1–2 mm). Also common are small scales and spangles; less common are nuggets, crystals, wires, leaves, tufts and hairs and arborescent, reticulated, dendritic, filiform, mossy and spongy forms. Extremely finely divided gold (flour, float or skim gold), the bane of the placer miner and geochemist, is common in some placer districts. Some of the largest natural specimens of the metal have come from placers, frequently of the eluvial type. From these have come some of the largest nuggets of gold won by man. The 'Welcome Stranger' found by accident in a cart rut in the eluvium near Ballarat, Victoria, Australia weighed 2516 oz troy and the 'Blanch Barkley' also from Victoria weighed 1743 oz. The Carson Hill nugget found in California topped the scales at 1296 oz. More generally the gold of placers occurs in small flattened scales or grains averaging a few millimetres in diameter or as moderately finely divided particles known as dust averaging a few tenths of a millimetre in diameter. The fineness of placer gold varies greatly and has been studied by many investigators. A survey of the literature indicates that for most placer deposits the fineness ranges from 500 to 999, that is from electrum to nearly pure metal; most placer gold is above 850 fine. The other elements in placer gold are mainly silver, copper and iron (*see also* Chapter II). Vein gold generally has a fineness that ranges from about 500 to 900. There are innumerable references in the literature that refer to the fact that in any particular district the fineness of the placer gold will usually be higher than that in the veins from which the metal presumably came. Lindgren (1933) for instance states that in California the vein gold averages 850 fine whereas the transported placer gold in the Tertiary channels averages 930 to 950. Similarly, the placer gold of Manhattan, Nevada is finer than the lode gold (Ferguson, 1917). In the Carolinas and Georgia the placers and eluvial deposits commonly have gold with a fineness greater than 900 whereas the vein gold is generally much less than this value, in places being as low as 500. Distance travelled and size of the placer gold also seem to be factors in the fineness, the further from the source and the smaller the size of the particles the higher the fineness. These features are difficult to assess in the writer's experience, but they are evidently true according to a number of placer miners who were consulted on the matter. Certainly, flour gold generally has a high fineness. Hill (1915) and Hite (1933*a,b*) gave an average of about 950 as the fineness of the Snake River flour gold. There is much evidence to suggest that most of the gold in gossans and oxidized zones is of greater fineness than that in the primary ores. Fisher (1945), who made a worldwide study of the fineness of gold in various environments, concluded that gold in the oxidized zone is nearly always higher in grade than the primary gold, and this difference varies largely according to chemical conditions and the facilities that exist for removal of the silver in solution, the gold being redeposited close by, or merely left enriched by the removal of the silver. Under conditions suitable for taking gold into solution the gold redeposited, if the concentration of silver remains high, will not necessarily be of high fineness. He went on to state that very high gold fineness is usually the result of oxidation under conditions favourable for the complete

removal of silver, e.g., Mt. Morgan, Queensland, and that high fineness is often associated with the oxidation of telluride ores, e.g., Fiji and Cripple Creek, Colorado. Finally, he noted that alluvial gold, being derived from the oxidized ore, is of higher quality (fineness) than the average of the vein gold, and shows an increase in value downstream, due to surface refining action, as the size of the grains decreases.

In Yukon Territory and at Yellowknife, Northwest Territories some of the supergene gold examined by the writer is of great fineness (990), whereas the primary gold is in the range 900 to 950. Another feature of placer gold, first examined by McConnell (1907) in the Klondike, is that the outer parts of nuggets commonly have a higher fineness compared with the inner parts. The writer has examined some of McConnell's samples and others from a number of placers by electron probe, and found that the rim of greater fineness is rarely more than 0.03 mm (30 μ). A rim of relatively pure gold on nuggets seems to be rather general since Johnston and Uglow (1926) found the effect on the gold of the Cariboo of British Columbia; Fisher (1935) in the Morobe goldfield, New Guinea; and Petrovskaya and Fastalovich (1955) in much of the placer gold of U.S.S.R. Ramdohr (1965) also noted the phenomenon in Rheingold; Mustart (1965) in much of the placer gold of Yukon; and Desborough *et al.* (1970) in placer gold from various parts of the United States. Stumpfl and Clark (1965), however, rightly point out that the rind of purer gold on nuggets, flakes and dust is not always present. They note its absence in gold-platinoid concentrates from placers in southeast Borneo. The rim effect has generally been interpreted as the result of solution of silver, but the writer is of the opinion that it represents the precipitation of gold (and some silver) on gold nuclei. An argument in favour of the latter process is given in a later section on the origin of placers.

It should be noted that the rim effect in placer gold evidently begins during oxidation of the primary deposits, because it is found in gold in the oxidized zones of auriferous deposits and in eluvial deposits. Gold with enriched rims has been observed by a number of investigators in near-surface shallow deposits (mainly Tertiary) in a number of places, and the phenomenon is noticeable in some kurokô ores in Japan. Enriched rims are rare in deep-seated gold deposits in the writer's experience. In the oxidized ores and placer deposits the rims are generally much wider and better developed than in the primary deposits.

Placer gold commonly contains many inclusions usually small in size (0.005–0.50mm). The inclusions as seen in polished sections and under the microprobe include various sulphides, particularly pyrite, galena and chalcopyrite; also arsenopyrite, various sulphosalts, tellurides and quartz, sericite, rutile, etc., most of the minerals normally found in gold deposits. Many of these minerals are probably original constituents of the gold nuggets since they are commonly near the core of the gold particles. Some often appear as though they were the nuclei about which gold precipitated in the oxidized zones of the deposits. These inclusions should receive detailed study during geochemical prospecting surveys since one might be able to divine the exact source of the gold, especially if material from a few primary sources in the district under investigation is available for control purposes.

Other internal features of placer gold have been extensively studied. In the classic paper by Fisher (1935) the high grade rims referred to above and the general granular structure of the nuggets and flakes are mentioned. Petrovskaya and Fastalovich (1955), who have carried out a long term study of the placer gold of U.S.S.R., also note the nearly universal occurrence of high grade rims and reentrants on gold nuggets and flakes and found that these have a fine-grained polyhedral texture rather than a laminated one as thought by the earlier investigators. They also noted that much of the placer gold retains the internal granular structure and other microscopic features of the primary deposits of gold, but found that much placer gold, especially that held in placers for long periods of time, exhibits marked internal deformation and recrystallization textures. They also observed the presence of intergranular stringers of gold in nuggets and flakes. These were found in the placer gold and in the gold from oxidized zones of auriferous deposits but not in the gold from the primary ores.

The external form, colour, lustre and other features of placer gold visible to the naked eye are commonly characteristic. The miners and bankers in a placer district can generally tell from which creek the gold originated simply by visual inspection. It is said that one banker in Dawson in its heyday could tell precisely from which of the many creeks and gulches of the Klondike a specimen of gold originated, and I am inclined to believe the story having seen gold from all of the creeks either in the old gold room of Yukon Consolidated, Ltd. or in the National Mineral Collection of Canada.

Placer gold usually has an entirely different appearance from that found in veins and other deposits. The high lustre of the vein gold is replaced by a subdued lustre in the placer, due it appears to incipient crystallization on the surface. Some varieties of gold are coloured due to a number of reasons. Black and deep brown gold are commonly due to the presence of coatings of manganese and iron oxides or manganese and iron humates. Alternatively some black and brownish gold is due to a coating of very finely divided colloidal or microcrystalline gold. Whitish and dull greyish varieties of gold owe their colour generally to thin coatings of calcium carbonate, colloidal silica or fine-grained (colloidal) clay. In most placers the gold has the characteristic golden colour, but in others foreign coatings are quite common. 'Black gold' (ouro preto) is very characteristic of a number of placer districts in Brazil and other tropical areas.

The writer has either seen, heard about or read about practically every form of gold in placers and the closely associated oxidized zones of gold deposits. The most common forms are scales, plates and nuggets. In addition some placers contain crystals of gold, hopper-shaped crystallized particles, masses having filiform, reticulated and dendritic shapes and films, wire and mossy gold. The growth forms of large nuggets are commonly fairly regular, many looking like nuts or potatoes; other nuggets are highly irregular and gnarled in shape. Some partly enclose vein quartz fragments or rounded quartz and other types of pebbles and mineral fragments. Some nuggets have unusual growth proturberances that may appear as distorted crystals. A few nuggets have been found that have secondary crystalline outlines.

Various terms are used in the following descriptions to give a semi-quantitative estimation of the size of the gold

particles in placers. A 'colour' has no exact meaning; it is used by placer miners to refer to a small piece of gold that usually varies from 1/16 in. (1.5mm) upward. Estimation of the value of gold colours in a pan and hence of the value of placer grounds can only be gained by experience since the value varies with the size of the colours, their thickness, and their purity. A good estimate can be made by picking out the colours from a number of pans, weighting them on a small pocket microbalance, and calculating the value per yard. There are about 50 pans (18 in. diameter) per yard. Coarse gold (nuggets) is usually considered to be that which remains on a 10-mesh screen (>1.5mm); medium gold will pass a 10-mesh screen and be held on a 20-mesh screen; fine gold passes a 20-mesh screen and is held on a 40-mesh screen (value 1 colour per cent); and very fine gold passes a 40-mesh screen. Flour gold is very much finer than the last; in some places 500 to 1000 colours or more are required to equal the value of a cent!

In general it can be said that the farther that gold has travelled from its source the smaller in size the particles become. Near the source the gold particles are usually rough and relatively large; with distance the particles become smooth, much rounded and smaller. The particles also apparently lose mass with distance from the source. Antweiler and Lindsey (U.S. Geol. Surv. Circ. 622, 1969*b*, p. 6) noted that particles of the Snake River gold, near its source below the outcrops of auriferous conglomerates in Jackson Hole, with diameters approximating 0.125 mm, averaged 10µg in weight, whereas near Twin Falls, Idaho, some 180 mi downstream, particles of the same diameter weighed only half as much. The cause of the mass loss was not discussed.

To summarize this introduction we can say that for the development of placers of any type, four requisites are necessary:

1. The occurrence of gold in deposits, in widespread quartz veins and blows or in a disseminated form in pyritic shales or other country rocks.

2. A fairly long period of deep secular chemical and mechanical weathering on a surface of submature to mature topography, during which time the gold is set free from the deposits or country rocks.

3. Concentration of the gold by some agency, generally water.

4. Absence of extensive glaciation: glaciation does not entirely preclude the occurrence of placers since both eluvial and alluvial types of placers may be overriden and little disturbed by the glaciers in some cases and buried by their deposits of till, clay, etc. in others. Such placers occur in British Columbia, Alaska, Yukon, Quebec and the Lena district of U.S.S.R.

Country kindly to the occurrence of extensive placers is readily recognized. The topography is subdued and marked by broad, often terraced entrenched valleys and rounded deeply weathered hills commonly with nearly accordant summit levels. Few extensive placers are found in terrains marked by sharp alpine features and high gradient V-shaped valleys; similarly, excessively flat terrains far from mountain systems and their foothills yield few productive placers. Like all generalities in geomorphology and geology there are some exceptions to these observations.

The literature on placers is voluminous. Works containing discussions of general principles include those by Liversidge (1893*c*), Weatherbe (1907), Crane (1908), Tyrrell (1912), Longridge (1913), MacKay (1921), Raeburn and Milner (1927), Park (1927), Idriess (1933), Boericke (1933), Lindgren (1933), Gardner and Johnson (1934–1935), Fisher (1935), Crampton (1937), Bilibin (1938), Mertie (1940), Averill (1946), Grutterink (1950), Bateman (1950), Shilo (1956, 1960), Gorbunov (1959), Griffith (1960), Harrison (1962), Vasyunina (1963), Barkovskaya (1963), Trofimov (1964), Fayzullin (1968), Ivenson *et al.* (1969), Shilo and Shumilov (1970, 1976), Valpeter and Davidenko (1970), Romanowitz *et al.* (1970), Volarovich and Shokhor (1970), Kartashov (1971), West (1971), Kazakevich (1972, 1976), Sigov *et al.* (1972), Wells (1973), Denisov (1973), Tayurskii (1973), Kogen *et al.* (1974), Kushnarev and Kashcheev (1974), Tishchenko and Tishchenko (1974), Segerstrom and Ryberg (1974), McLellan *et al.* (1974), Saks (1974, 1975), Vorob'yev and Krapivner (1975), Clift (1975), Prokuronov (1975), Smirnov (1976), Kisterov (1976), Fricker (1976) and Zhelnin and Travin (1976).

The references by Gardner and Johnson (1934–1935), Romanowitz *et al.* (1970), West (1971), Wells (1973) and Fricker (1976) contain up-to-date summaries of the procedures for the prospecting, sampling, evaluation and working of placers. It need hardly be emphasized here that the processes involved in the concentration of gold in placers are extremely complex, and the conditions that control them so variable that even under the most favourable circumstances observable, it is generally not possible to estimate the potential of a placer even roughly without detailed sampling, either by pitting, test holes, or deep cuts or trenches.

The distribution of the principal eluvial and alluvial auriferous placer districts of the world is shown in Figure 67. Many of the districts have long been exhausted, some before the beginning of the Christian Era.

Papers and other works describing placer areas in various parts of the world and methods whereby they are worked are included in the Selected bibliography.

Eluvial placers

Eluvial placers[17] are formed in the weathered residuum over or in the immediate vicinity of primary gold deposits of all types. Commonly these placers develop downhill from the outcrop of the primary deposits (Fig. 68). Some are of the nature of talus deposits or fanglomerate facies on mountain slopes or in the debris at the foots of hills and mountains. An unusual type is formed in the depressions of deeply and irregularly weathered terrane (karst topography). Eluvial placers are closely associated with alluvial placers in many districts, the former giving birth to the latter in many places.

The principal mechanism for the concentration of heavy minerals in eluvial placers is the winnowing action of gravity and downhill creep, the latter being essentially dependent on

[17] Some geologists, particularly in U.S.S.R., classify these placers as eluvial, deluvial and proluvial. By this classification eluvial placers are those whose outlines coincide more or less with those of the primary deposits. Deluvial (scree or talus) placers are those whose upper limit is at or near the primary source and whose downhill front lies at the foot of a slope. Proluvial placers form in the disintegrated debris at the foot of hills or mountains.

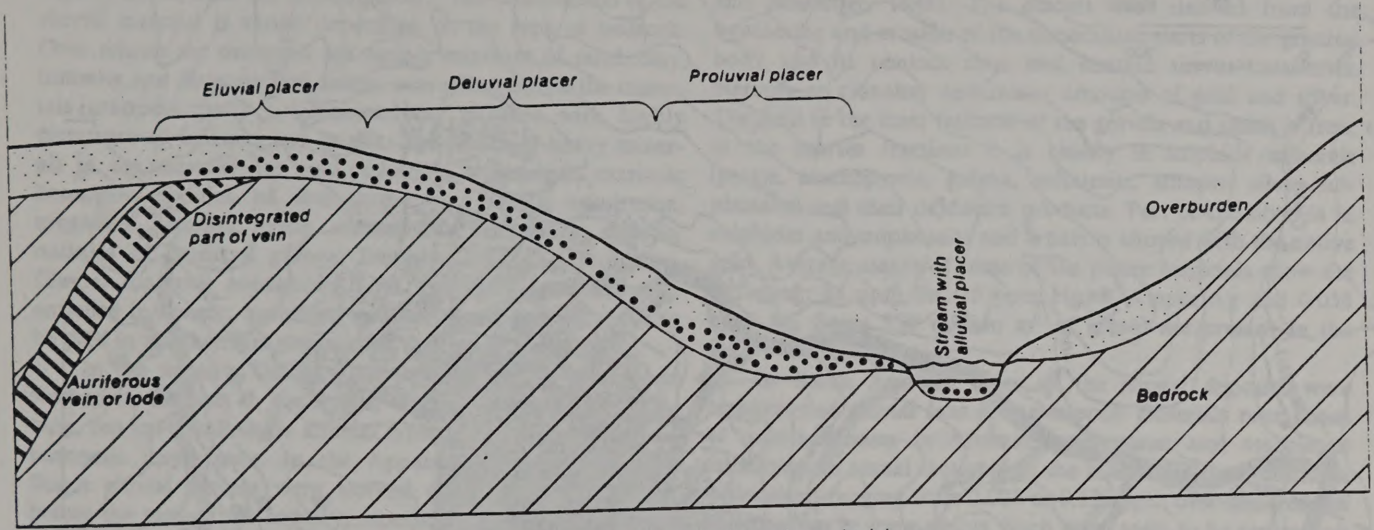


Figure 68. Sketch illustrating outcrop of a gold-quartz vein supplying material to form eluvial and alluvial placers. By one terminology, an eluvial placer embraces all materials not transported by streams. By another, only the placer materials over weathered and disintegrated deposits constitute an eluvial placer; downslope and other placer materials are classified as deluvial and proluvial as shown.

the angle of slope or gradient where the placers are formed on the sides of hills or mountains. Secondary factors include the thickness of the slope materials (scree, talus, residuum), the size and specific gravity of the weathered particles in the residuum, the coefficients of thermal expansion and contraction of residual particles, the coefficient of friction, the movement of ice and snow (glaciers) and the annual and daily variation of temperature. The last governs the freezing and thawing of the residuum in temperate zones, and is particularly important in permafrost regions where solifluction is widespread. Continuous downhill creep appears to be the main way by which gold reaches gulches and creeks. Moving water plays only a small part in the concentration of heavy minerals in eluvial placers, and it is common to find that these placers are largely independent of water courses either in the detritus or on the surface.

Auriferous eluvial placers commonly exhibit considerable gradation and variation in the content and nature of their gold and associated heavy and light minerals. Near the primary deposits the particles of gold are invariably larger and commonly of less fineness than those farther down the dispersion fans and trains on the slopes. During the downward creep of minerals and rock fragments on slopes the lighter fractions tend to gravitate to the top layers and move relatively faster downhill than the heavier fractions. This is why one commonly finds a greater concentration of light minerals and rock fragments such as quartz floats at the foots of hills and mountains rather than farther uphill and at the outcrop of auriferous quartz veins. Finally, nonuniformity and erratic behaviour commonly prevail in the continuity and value of the pay streaks in most eluvial placers, a feature that dictates that most of these deposits must be carefully pitted, trenched, drilled and evaluated before exploitation.

Eluvial gold placers were widely worked in many parts of the world prior to the turn of the century. Most were residual blanketlike deposits developed on a relatively flat terrain, or fanlike accumulations formed on gentle slopes. The materials of these deposits varied greatly depending principally on the types of bedrocks and deposits and the type and degree of weathering. Most had a high iron (lateritic) content although some were relatively highly enriched in aluminum (bauxite or clay). Most consisted of a residuum of quartz, sand, clay, iron oxides, aluminum oxides and manganese oxides in which the particles of gold, commonly with a number of other chemically resistant heavy minerals, were concentrated on or near the bedrock. Some of these deposits are highly cemented with clay, limonite or caliche (carbonates); more rarely by silica. Most eluvial placers are low grade in gold, but some are of enormous yardage (tonnage). Much of the gold in eluvial placers is rough and irregular in form, and the fineness is commonly only slightly higher than that in the primary deposits. Eluvial placers are known for their large nuggets or masses of gold, some weighing several hundred to a thousand ounces or more; alluvial placers only rarely contain large nuggets ($>$100 oz). Some eluvial placers yield silver, lead, cassiterite, cinnabar, diamonds and platinoids in addition to gold. A few descriptions of typical examples of eluvial deposits follow.

Extensive eluvial placers are unknown in Canada, but there are a number of small examples mainly in Yukon that are of interest from an academic viewpoint. In the Dublin Gulch area, Yukon (Boyle, 1965*a*), small eluvial placers are developed on the sides of the hills above the economic gold placer in the gulch (Fig. 69). The eluvial placers contain mainly scheelite with small amounts of wolframite and gold, these minerals being derived from a complex of scheelite-

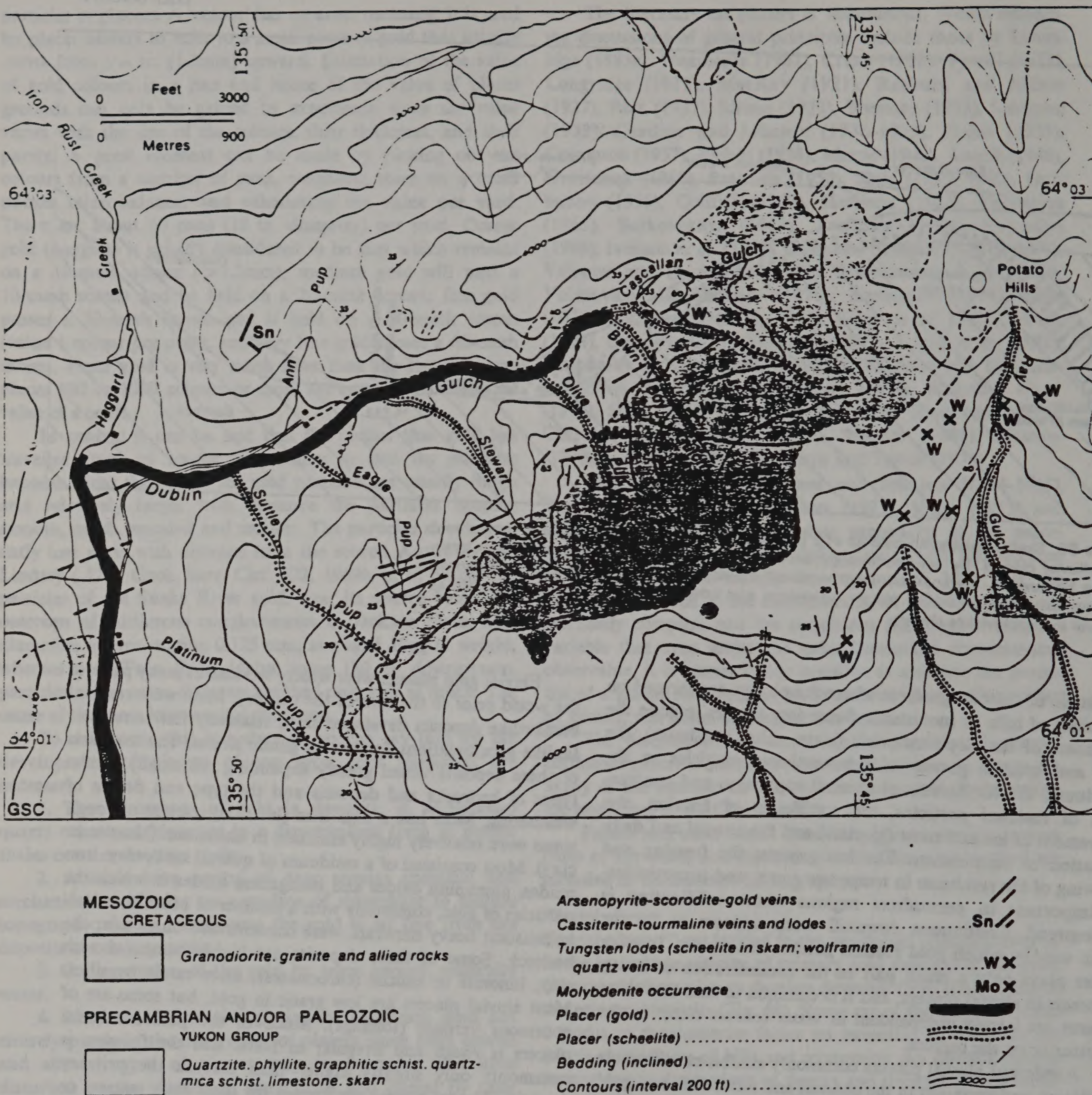


Figure 69. Generalized geology of Dublin Gulch area, Yukon showing gold and scheelite gulch placers (after Boyle, 1965a).

bearing skarn, quartz-wolframite veins and northeast striking quartz-arsenopyrite-pyrite-scorodite-gold veins, all lying slightly uphill from the eluvial placers. The constitution of the eluvial material is varied depending on the type of bedrock. Over schists the materials are mainly mixtures of sand, clay, limonite and disintegrated schist; over granodiorite the materials are principally a limonitic clay mixture with highly disintegrated feldspar and quartz. The principal heavy minerals in the eluvium are gold, scheelite, wolframite, extrinsic hematitic nodules of iron-formation, garnet, cassiterite, magnetite, bismuth, pyrite, arsenopyrite, jamesonite, siderite, nodules of oxidized galena, limonite, wad and scorodite. Eluvial material containing gold and scheelite is greatly enriched in arsenic, antimony and tungsten a feature that can be used in geochemical prospecting for this type of deposit.

In the western United States eluvial placers have been worked in the past in California, Oregon, Nevada and Montana, but few rivalled the alluvial types in these states in their economic importance. In the Appalachians of the United States eluvial placers were worked on a fairly large scale before the turn of the century, especially in Georgia, (Becker, 1895; Pardee and Park, 1948; and Lesure, 1971). There, in the Dahlonega gold belt in Lumpkin County, the bedrock is a complex of mica schist, mica gneiss, quartzite, amphibolite, migmatites and granite. The schists contain abundant gold-bearing quartz veins and stringers of various ages. All rocks and deposits are deeply weathered forming at the surface what has been called a saprolite by Becker (1895). Saprolite is in reality only a form of laterite; it is brownish in colour, composed of soft, earthy, clay-rich materials derived from thoroughly decomposed schist and other metamorphic rocks. Zones of disorganized and disintegrated quartz veining in the saprolitic schists were mined by deep open cuts on a large scale near Dahlonega, the gold being won partly by washing and partly by crushing of the quartz in stamp mills. The enrichment in the saprolite was evidently due to considerable leaching and precipitation of gold as suggested by the geochemical results quoted by Lesure (1971), who found up to 2.9 ppm in freshly precipitated limonite. The gold distribution in the fresh vein quartz is low and spotty, and the primary deposits are apparently not attractive mining ventures.

Certain gossans on stockworks or massive sulphide deposits and their attendant eluvial deposits have been mined or are being mined for gold. At the Greenhorn Mine in Shasta County, California an auriferous gossan was worked profitably for many years (Huttl, 1940). The gossan was developed on a copper sulphide orebody and consisted of a mass of limonitic material of varying thickness containing localized zones of rhyolite. The principal values in the gossan were gold and silver with some flakes of native copper. The gold particles were extremely fine, almost microscopic. The combined gold and silver content averaged about 0.20 oz/ton. Similar gossans have been mined in many countries, e.g., Spain (Rio Tinto), Australia (Mt. Morgan) and elsewhere. Some of these are described in more detail in Chapter IV.

Eluvial deposits have been worked in many places in the lateritic residuum in Mexico, Central America, Cuba, Haiti, Puerto Rico and the countries of South America.

In Mexico eluvial-alluvial placers in three large fans of gravels downslope from a mineralized porphyritic granite stock and its limestone contact zone are found near Guadalcázar in the central part of the State of San Luis Potosi (Fries and Schmitter, 1948). The placers were derived from the weathering and erosion of the mineralized parts of the granitic body and its contact zone and contain tin as cassiterite, mercury as cinnabar and minor amounts of gold and silver. The gold in the finer fraction of the gravels and sands is free; in the coarser fractions it is chiefly in sulphide minerals (pyrite, arsenopyrite, galena, sphalerite, stibnite, silver sulphosalts) and their oxidation products. Part of the silver is in sulphides and sulphosalts and a part is alloyed with the native gold. Average assays of some of the placer materials show the following: 84 ppm Sn, 15 ppm Hg, 9.44 ppm Ag and 0.058 ppm Au. Some 550 million m^3 of gravel are present in the fans.

In South America most of the eluvial deposits were blankets and eluvial fans in the lateritic materials near zones of quartz veining in Archean greenstones and associated sediments. In actual fact gold in the laterites of the Guianas is widespread ranging from 0.003 to 0.135 ppm over hundreds of square miles in some places. Such areas seem to be centred on auriferous Precambrian greenstone and sedimentary belts intruded by granitic rocks and porphyries. During weathering, therefore, gold has apparently been concentrated in the laterites both from the rocks and from epigenetic gold-bearing deposits. Extensive natural winnowing of the laterites has evidently given rise to the important alluvial placers, which are a feature of the Guianas. One interesting deposit at Omai, Guiana (British Guiana) was a deeply weathered aplite dyke highly decomposed to a depth of more than 100 ft (Maclaren, 1908). The primary dyke material was shot through by slightly auriferous quartz stringers and heavily pyritized, the pyrite being auriferous. The decomposed material consisted essentially of limonite-stained sand and clay, in which well crystallized free gold was abundant.

Other interesting types of eluvial deposits occur in Brazil mainly in the Quadrilátero Ferrífero, Minas Gerais. All of these are associated with gold-bearing iron-formations (itabirites). General descriptions of these deposits are given by Derby (1884, 1903), Bensusan (1929), De Oliveira (1930, 1932), Gair (1962) and Dorr (1969) (*see also* p. 304 of this chapter). One type of secondary deposit, not necessarily eluvial, but often grading into eluvial deposits is locally known as "jacutinga" (De Oliveira, 1932). The jacutinga occurs as thin (inches to a few feet) lines or bands in itabirite and is a decomposition, or more accurately a surficial chemical disintegration product of the iron-formation. It is composed essentially of powdery ferric oxide (limonite and hematite) manganese oxides, clay minerals and talc in which nuggets, plates and threads of native gold are present. Evidently the gold originated by chemical solution and reprecipitation of the metal, from low grade gold-bearing iron-formation. From the descriptions, some of the jacutinga were rich, up to 0.5 oz Au/ton or more, and some were mined to depths of 700 ft. Some of the gold was rich in palladium.

The other type of deposit associated with iron-formations in Brazil is the gold-bearing 'Tapanhoancanga' or 'Canga' for short. This is an irregular layer or blanket up to 10 ft or more thick of limonite-cemented fragments of iron-formation (itabirite). It is commonly developed on all iron-

formations throughout the world that are extensively oxidized. Much of the limonite appears to derive from the oxidation of iron silicates and pyrite in the iron-formation. Where the iron-formations are also gold-bearing the canga is commonly enriched in gold, the metal occurring in small flakes, wires, specks and also in a submicroscopic form associated in some manner or other with the limonite. In the past, deposits of this type were worked in a small way in many parts of the ferriferous quadrangle in Minas Gerais.

Eluvial placers have not been important sources of gold in Europe although they have been worked in the past, some by the Romans, in Portugal, Spain, France, Switzerland, Germany and Romania. Important eluvial deposits were once worked in the Berezovsk, Kochkar and other districts in the U.S.S.R. In Africa a few eluvial placers have been worked mainly in Egypt, Sudan, Ethiopia, Zaire, Ghana and elsewhere in the western gold belt of Africa. Most were small but have mothered extensive alluvial placers in a number of countries. It is of interest to note that the great Rand deposits of South Africa gave rise neither to extensive eluvial nor to alluvial deposits, a feature that is difficult to explain. The reason for the paucity of placer gold associated with these enormous deposits probably has much to do with the size of the gold in the reefs (50 μ) and the presence of abundant pyrite, which weathers readily to give soluble iron salts and colloidal hydrous iron oxides. Both of these features would tend to render the gold soluble or mobile as an absorbed component, thus ensuring its broad dispersion.

Eluvial gold was relatively common in Australia before the turn of the century, but most of these placers are now exhausted. Fabulous nuggets were found in some districts in the eluvium or weathered rubble of the oxidized deposits. At Big Nugget Hill in the Hargraves goldfield of New South Wales a nugget weighing 106 lb was found in the eluvium by an aboriginal shepherd. At Lucknow, also in New South Wales, the eluvium was of great richness at and near the outcrop of the veins. In Victoria even larger nuggets were found in the eluvium and disintegrated country rock near the veins of Ballarat, Tarnagulla and other places, including the Welcome Stranger (2516 oz), Welcome (2195 oz), Blanch Barkley (1743 oz), Canadian (1319 oz), Dunolly (1364 oz) and Sarah Sands (755 oz). In Western Australia the eluvial placers were small, rich and soon worked out in the Pilbara, Kimberly, Coolgardie, Kalgoorlie and other Yilgarn fields.

In Asia eluvial gold has been won from soils and weathered debris near outcroppings of gold in Japan, China, Burma, India, U.S.S.R. and elsewhere. Most of these deposits were small and have long been exhausted. One type of eluvial deposit in U.S.S.R. is of considerable interest both economically and scientifically – the Kuranakh type. The Kuranakh gold deposits found in 1959–1962 are mainly in Lower Cambrian bituminous limestones and dolomites in the Aldan anteclise of southern Yakutia (Razin and Rozhkov, 1963, 1966; Kazarinov, 1969; Borodaevskaya and Rozhkov, 1974). There is considerable similarity between the primary gold occurrences and those described in this chapter for the disseminated deposits of north-central Nevada (Carlin, Cortez, etc.). The primary occurrences in the Kuranakh district are mainly potash feldspar (adularia) and quartz metasomatites with about 6 ppm gold occurring as irregular replacements of the limestones and dolomites in places in stratiform bodies up to 3 km in length, 300 to 500 m in breadth and 5 to 10 m in thickness. The gold in the adularia zones is very finely-divided (2–5 μ) and mainly in pyrite. Thin kersantite dykes and sills of late Mesozoic age are also accompanied by adularization along their contacts both in the Cambrian strata and in Lower Jurassic sandstones, and these igneous rocks have a comparable tenor of gold within the ore zones. Oxidation of the primary gold-bearing zones and karstification of the Lower Cambrian limestones in post-Jurassic and Recent times has given rise to economic eluvial deposits consisting essentially of highly weathered material composed mainly of oxidized ore, limonite, clay and sandy materials in the karst cavities (Fig. 70). The gold in the eluvial material occurs as very finely-divided flakes ranging in size from 5 to 20 μ. The ratio of concentration is not given, but presumably it is about 2 making the grade of the deposits about 12 ppm (0.35 oz Au/ton).

In Luzon, Philippines, eluvial placers are common and were extensively mined in the past. Gilbert (1940) describes a number of these from the Paracale-Mambulao district where the primary auriferous material is mainly veins and stockworks of quartz and pyritic quartz veins in granite intruding serpentinites. The auriferous eluvial material consisted mainly of weathered debris and disintegrated bedrock carrying free gold.

Certain eluvial auriferous deposits take the form of talus accumulations. A typical example of this type of placer occurs near the Belaya Gora deposit in the Lower Amur region of U.S.S.R. (Borodaevskaya and Rozhkov, 1974). The bedrock deposit occurs near the summit of Belaya Gora in a strongly fractured, kaolinized and silicified Oligocene volcanic neck. The gold is mainly free and accompanied by about 0.5 per cent combined pyrite, arsenopyrite, sphalerite and argentiferous sulphosalts. The talus placer associated with the primary deposit lies below the deposit on a gentle slope and averages some 5–6 m in thickness. In composition the talus material is largely variegated clay and sandy clay and oxidized rubble with a heavy mineral suite composed mainly of limonite, magnetite, ilmenite, chromite, epidote and small amounts of zircon, sulphides, manganese oxides and other minerals. The gold is disseminated through the talus material and also forms short streaks and enriched lenses. The gold in the talus is not much different from that in the bedrock. The gold grains range in size from 0.05 to 1.5 mm. Some adsorbed and very finely divided gold also occurs in concretions of kaolinite and quartz. Downhill from the talus placer there is a colluvial and alluvial placer system associated with springs and the river fed by the springs.

Alluvial placers

Alluvial placers are those formed in present and past water courses in gulches, creeks, rivers, flood plains and deltas. Reworking of some of these deposits together with others formed as a result of sedimentation or glacial processes by wave action may yield beach placers, which are treated separately.

Alluvial placers have been worked since ancient times in practically every country (Fig. 67) and have produced probably about one-quarter of man's store of gold. If we include the

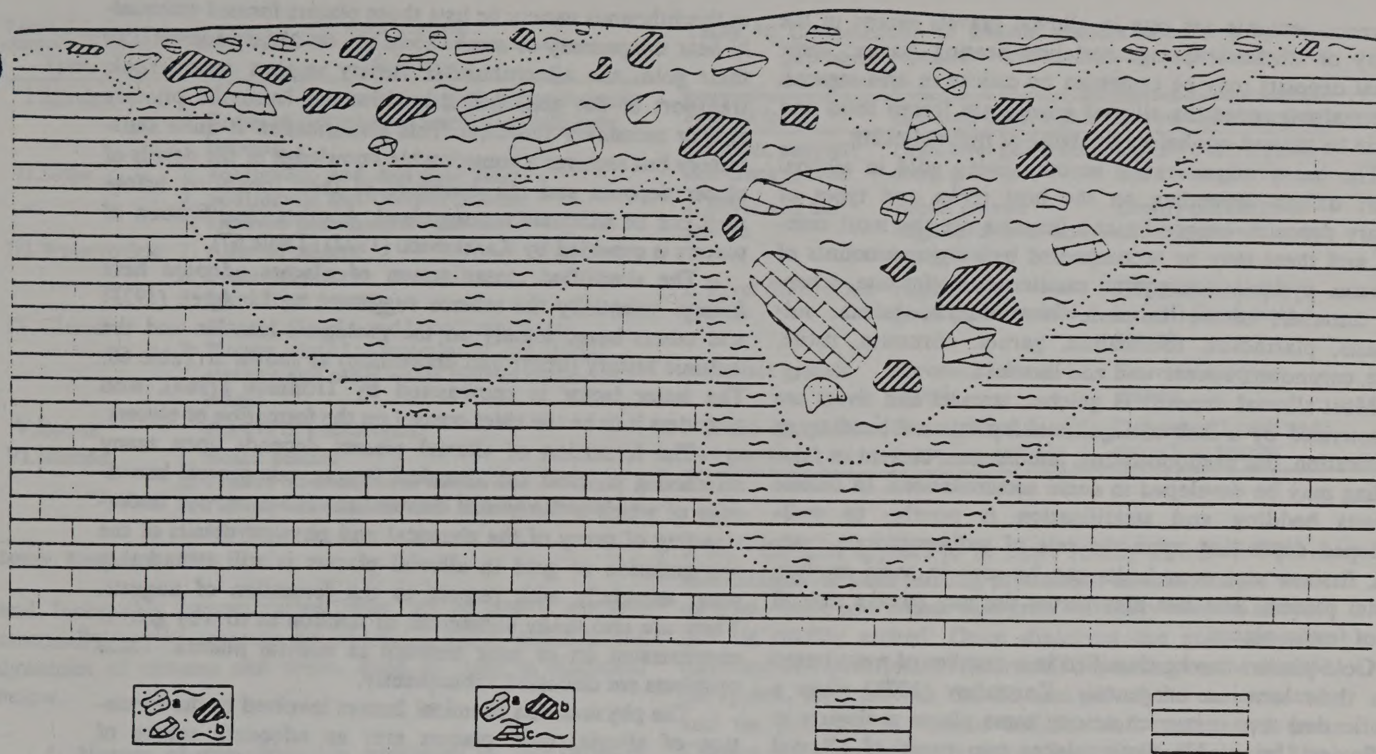


Figure 70. Schematic cross-section of an auriferous karst, Kuranakh district, southern Yakutia, U.S.S.R. (after Razin and Rozhkov, 1963). (No scale given on original.)

Witwatersrand deposits as fossil alluvial placers, the amount of gold produced from these types of placers probably approaches two-thirds of man's store of the precious metal.

Alluvial placers can be classified into two general categories – modern and fossil. The distinction between the two is commonly difficult to make in the field. Placers formed in present day water courses and most of those of Pleistocene and Tertiary age fall into the modern category. Those of greater age, commonly buried deeply by superincumbent sediments or volcanics and generally lithified we shall call fossil. Fossil placers occur throughout the geological column and are described subsequently.

There is an enormous amount of literature on alluvial gold placers, dealing with descriptions of the placer fields, the principles of placer formation and the methods of panning, rocking, sluicing, hydraulicking and dredging. The interested reader is referred to the general references mentioned in the introduction to the section on placers.

There are some general characteristics of alluvial gold placers that will serve as a basis for the descriptions and discussion that follow. These include: alluvial placer gold in pay streaks near its source is invariably coarse and is found in the lower layers of the alluvium either on bedrock, in a zone a few feet above the bedrock or in crevices, fractures, etc. in the bedrock within a few feet of the surface. An exception to this is where the so called 'false bottoms' or 'false bedrocks' occur in thick beds of alluvium. These false bottoms may be clay layers (hard pans) within the gravels, compacted sands or more rarely limonite-cemented sands and gravels (conglomerates). Alluvial placer gold far from its source is generally finely divided, and while part of the gold may be on or near bedrock or false bottoms most of it is dispersed throughout great thicknesses of the sediments. This is especially true of extensive flood plain and deltaic deposits. Two other features are characteristic of most alluvial gold placers. The first, as intimated above, is that the further from the source the more finely divided the gold, and the second is that placer gold is finer in value than its source lode gold and that with increasing distance from the source the finer is the gold. These two statements are general. The first is invariable, but there are exceptions to the second in some districts.

Alluvial placers are composed of loose unconsolidated gravels and sands that are commonly relatively clean. The terms 'white channel gravels', 'white leads' and 'white bars' reflect the latter circumstances with respect to quartz. In places, however, the pebbles and the gold may be coated with limonite, wad and other precipitates. Some alluvial gravels and sands are heavily impregnated with limonite and wad forming the 'cement gravels' or 'cangalli' of the alluvial miner. Such gravels invariably occur where the primary deposits and wall rocks are rich in pyrite, siderite, chlorite and other iron- and manganese-bearing minerals. Secondary siliceous and

calcareous cements are rare in alluvial gravels except in the vicinity of siliceous springs and lime-bearing waters; some alluvial deposits may be cemented by caliche in arid regions. In permafrost zones the alluvial gravels are frozen solid and have to be thawed out before dredging or hydraulicking.

The heavy mineral suite accompanying gold in alluvial placers differs depending on the host rocks and types of primary deposits. Magnetite and ilmenite are the most common, and these may be accompanied by varying amounts of monazite, pyrite, arsenopyrite, cassiterite, wolframite, scheelite, cinnabar, native bismuth, bismuthinite, galena, sulphosalts, platinoids, tourmaline, garnet, chromite, rutile, barite, corundum, zircon, wad and limonite.

Most alluvial deposits in gulches, streams and rivers are characterized by a lack of regular and persistent bedding or stratification, but pseudobedding, laminations, current or false bedding may be developed in some accumulations. In deltaic deposits bedding and stratification is poorly- to well-developed depending upon the rate of sedimentation. Fractures, fissures and even faults can be seen in some modern alluvial placers. The last may throw the pay streaks several tens of feet in places.

Gold placers can be classified in a number of ways based upon their location or genesis. Kartashov (1971) gives a classification that is current among some placer geologists in the Soviet Union. He distinguishes two types of alluvial placers – autochthonous and allochthonous (Table 66). In the authochthonous variety he lists those placers formed essentially near the primary or more rarely the secondary source(s) of their gold; the allochthonous variety implies considerable transport of the gold and deposition far from the primary and/or secondary source(s). This classification is quite satisfactory but requires a considerable knowledge of the details of placer deposits and the dynamics of their formation before they can be adequately categorized. Another classification of placers is provided by Kazakevich (1972) (Table 67).

The simplified classification of placers adopted here follows essentially the scheme suggested by Lindgren (1933) and others based mainly on the geological location and the tectonic history (uplift and depression) as shown in Table 68. The latter factor is emphasized by Trofimov (1964), who considers it to be the chief control on the formation of placers.

The formation of alluvial placers depends upon many interacting physical and chemical factors, our general knowledge of which is fairly well established. However, our understanding of many of the chemical and physical details of the accumulation of gold in alluvial placers is still remarkably poor, especially with respect to the formation of nuggets. There are also many differences of opinion as to why gold is concentrated on or near bedrock in alluvial placers. These problems are discussed subsequently.

The physical and chemical factors involved in the formation of alluvial gold placers are: an adequate source of primary gold, favourable oxidation processes for its release

Table 66. Characteristics of the kinds of alluvial gold placers

Notes	Autochthonous: Bottom*	Autochthonous: Above-bottom*	Allochthonous
1. Are represented by	Channel, valley, terrace and watershed placers	Valley, terrace and watershed placers	Point-bar delta, river-plain, valley, terrace and water-shed placers
2. Occur	Adjacent to their gold-bearing sources		More or less far from their gold-bearing sources, being separated from them and from autochthonous placers by zones of dispersion of 'placer' minerals
3. Are concentrated	At the base of an instrative† or substrative† alluvium and in the crevices of the bedrock	At the base of a perstrative† alluvium and within constrative† strata, in the same parts of valleys as bottom placers	In surficial horizons of a perstrative alluvium and within constrative strata, downstream from autochthonous placers
4. Consist of mineral grains	Derived directly from gold-bearing sources or redeposited from older placers and not carried away by rivers from concentration zones		Brought by rivers into concentration zones
5. Accumulate during	Entire time of destruction of primary gold-bearing deposits, embracing, as a rule, many stages of river development	The last equilibrium and/or aggradation stages of river development	
6. An enclosing alluvium is formed during	Downcutting stages or transition from them to equilibrium stages	Equilibrium and/or aggradation stages	
7. An enclosing alluvium, being reworked during a downcutting stage, is	Not destroyed but displaced to the level of a new bedrock bottom	Displaced to the level of a new bedrock bottom and added to bottom placers	Completely destroyed
8. The mechanism of concentration of 'placer' minerals	Does not essentially depend upon the hydrodynamic properties of flowing water		Depends to a large extent upon the hydrodynamic properties of flowing water

Adopted from Kartashov (1971).
*These terms refer to the position of the pay streaks with respect to the bedrock.
†Instrative, perstrative and constrative correspond to the stages of river downcutting, dynamic equilibrium and aggradation. Refer to Figure 73 and the original article by Kartashov (1971).

Table 67. Genetic classification of gold placers

Type	Sub-type
I. Eluvial	1. Zones of weathering of gold-quartz deposits 2. Zones of oxidation of auriferous sulphide deposits
II. Slope	1. Solifluction and land slide placers 2. Solifluction and deluvial placers 3. Deluvial and landslide placers
III. Watercourse	1. Alluvial placers 2. Proluvial placers 3. Lacustrine placers
IV. Glacial	1. Placers of main and lateral moraines 2. Placers related to interglacial and glacial streams and other water courses of glacial origin
V. Eolian	Placers in eolian sands
VI. Marine	1. Beach placers 2. Placers on underwater slopes 3. Placers in near shore still water (lagoons, coves)

Source: Kazakevitch (1972).

Table 68. Classification of alluvial placers according to their geological location and tectonic history (uplift or depression)

Present form	Elevated equivalents	Depressed equivalents
1. Gulch, canyon and creek gravels	High level gulch and creek gravels	Deeply buried gulch and creek gravels
2. River and bar gravels	Bench gravels High level river and bar gravels	Deeply buried river and bar gravels
3. River flood-plain gravels	High level flood-plain gravels	Depressed (deeply buried) flood plain gravels
4. Deltaic gravels	Elevated deltaic gravels	Depressed deltaic gravels
5. Beach and shore line gravels	Elevated beach and shore line gravels	Depressed beach and shore line gravels

and favourable concentration sites, all of which depend essentially on the tectonic environment, which controls the dynamics of streams and rivers. Each of these is discussed below.

1. Nature of the oxidation processes in the primary gold-bearing deposits: several factors are involved, including climatic and groundwater conditions and the nature of the primary gold, gangue and wallrocks. These are discussed in detail in Chapter IV. The nature of the primary gold and the chemical system operating during oxidation of the deposits are two critical factors in the formation of alluvial placers.

The size of the primary gold particles is a major consideration. Where the particles of native gold in a primary deposit exceed 100 μ there appears to be relatively little solution or flotation of the gold in the natural waters. During oxidation processes the particles tend to go directly into the eluvium and creep from there into the alluvial placers. They may, however, accrete gold from the soil solutions and stream waters as they proceed on their journey.

In some districts, however, where the primary gold is extremely finely divided (<50 μ) or is a microscopic or lattice constituent of auriferous pyrite, arsenopyrite, tellurides, etc., alluvial placers are not formed to any extent even though other conditions are favourable. One striking example of this is the paucity of alluvial placers associated with the great Witwatersrand deposits. In such districts the gold does not seem to aggregate to form large particles. On the contrary it appears to be dispersed as flour gold or is dissolved and dispersed throughout the ground and surface water systems. In other districts the reverse situation exists, and alluvial deposits abound even though the gold is finely divided or a constituent of pyrite, etc. in the primary deposits. In these the gold accretes mainly in the oxidized zones and eluvial deposits to form relatively large particles, which ultimately find their way into the alluvial placers. To explain the difference in behaviour between the two extremes is not simple because of the complex chemical systems involved. In the writer's experience one important factor appears to be the purity of the finely divided primary gold. If its silver and/or copper content is high the gold appears to be more readily dissolved. This is understandable from etching procedures in microscopic work on gold – pure gold etches with great difficulty whereas alloyed gold is readily etched. Once dissolved the gold may be dispersed or reprecipitated (accreted) depending on the chemical system, especially on the Eh and pH of the environment and the presence of various colloids and precipitants (*see* Chapter IV). The relatively pure, finely divided gold, on the other hand, enters the erosion cycle chemically unaffected and is mechanically dispersed far and wide. Finely divided gold contained within pyrite and arsenopyrite and lattice gold in these minerals and in tellurides, aurostibite, etc. tends to accrete and form large particles, wires, etc. for reasons that are obscure but which may involve the solvent effects of thiosulphates, cyanides and ferric sulphate, interaction of the dissolved gold with natural reductants and various colloidal reactions. The surface characteristics of finely divided gold also seem to be important in its mechanical transport in water and probably also in its dissolution in oxidizing solutions. Some finely divided gold particles appear to have a relatively small contact angle due probably to films of organic substances, which permit air bubbles to attach themselves readily, a feature that promotes their easy flotation downstream. In the case of the Witwatersrand gold the film in some cases may be the hydrocarbon (thucholite) with which the gold is intimately associated. Films of all kinds, including both organic and inorganic substances, commonly inhibit the solution of finely divided gold in natural solutions. The most common inhibitors appear to be the various hydrous iron and manganese oxides and silica.

2. Nature of the bedrock or false bottoms: since alluvial gold is invariably found on or near the bedrock surface or on false bottoms the characteristics of these substrates are commonly decisive in the formation of alluvial placers. Few 'pay streaks' or 'runs of gold' occur on smooth bedrock bottoms for obvious reasons. The most favourable bedrocks are those that form natural riffles perpendicular to the stream or river course as in a gold sluice. Alternating thin beds of soft schist and hard quartzite or slate and quartzite are particularly favourable. It is a surprising fact, however, that in some placer areas bedrock riffles parallel to the direction of the stream flow are

more effective in trapping the gold than those crossing it at a large angle. Limestone bedrocks that are pitted and pinnacled provide particularly good settling places for gold, as do potholes and nests of boulders in some placer streams. Partly weathered schists and gneisses in which the leaves of the schist and bands of the gneiss are slightly separated are also ideal rocks for the concentration of alluvial gold. The little nuggets, flakes and specks of gold dust work their way into such rocks often to depths of 10 ft or more. Similar phenomena are noted where the bedrocks are sheared, fractured or intensely shattered. Limestones in karst terrane are also good traps for gold, the metal having worked its way down into the rock to depths of 30 or 40 ft in some placer areas. Mining of these zones of enriched bedrock is common in many alluvial gold districts. As for false bottoms, indurated clay and hardpan layers appear to be particularly favourable, although rough gravels and sands cemented by limonite, wad or carbonate provide suitable surfaces for the accumulation of gold in some districts. Compacted sands with rough surfaces are favourable in others. Some placer areas are characterized by stacked pay streaks due to alternating bands of clay, compacted sand, etc. in the sedimentary section.

3. Stream and river dynamics: this subject is complex involving hydrodynamics, sedimentation phenomena, geomorphology and isostasy (uplift and subsidence), subjects that can be treated only in a general manner here.

It is assumed that the reader is familiar with the general rudiments of the principles of sedimentation of falling bodies in static water. Briefly, the controlling factors involved are mainly the differences in specific gravity, size and shape of particles. It is axiomatic that of two spheres of the same weight but of different size, the smaller, with its lesser surface area, and hence lesser friction to water, sinks more rapidly in a static water medium. This is part of the reason why gold as small nuggets is commonly associated with quartz pebbles 1 in. or more in diameter.[33] Furthermore, the shape of particles is a factor – a spherical mineral has less surface area than a platy mineral of the same weight and hence sinks more rapidly. Finally, during jigging of bodies of variable specific gravity in water, those with the higher specific gravity sink to the lowest level. During this motion, round, shotty particles of gold will also find the bottom quicker than scaly or spongy particles.

We are, however, not generally dealing with static bodies of water in the formation of placers but with bodies of water that move and exhibit turbulent flow. The principal parameters in stream and river dynamics are the gradient and the volume of water. The gradient is determined by many secondary factors in an area but fundamentally by isostasy, which involves uplift (or depression) of one segment of the earth with respect to the other. The volume of water delivered to a stream or river, likewise, depends on many secondary factors involving the runoff and groundwater systems but primarily on the rainfall of the district. In a district with a pluvial climate once a gradient has been established stream and river systems develop that erode the land and yield a pattern of gulches, canyons and valleys.

The dynamics of the movement of sedimentary particles in streams and rivers are of interest in the formation of placers. The factors involved are many, including the velocity of the water, which depends essentially on the gradient; the degree of turbulent flow, the specific gravity of the minerals being transported and the nature of the stream or river bed. Two types of solid load transported by streams and rivers can be distinguished – the suspended load and the bed load. The suspended load is maintained in the body of water mainly by turbulence, although flotation of sedimentary particles attached to air bubbles may be important in places. The bed load consists of material that is rolled, pulled, slid or otherwise swept along the bed of the stream by traction or saltation (skipping of pebbles along the bottom in a series of leaps). In addition to these movements there is a constant creep of the sediment down gradient. This effect is small where the slope is minor, but may be marked where high gradients prevail. As an approximation it is said that the transporting power of a stream varies as the sixth power of the velocity. Calculations show that a stream running 2 mph (3 ft/sec) will carry a stone or nugget weighing about 3 oz; at 4 mph one weighing 45 lb; at 10 mph one weighing 1.5 tons; and so on. It is easy to understand how a spate of water moving down a gulch at 20 mph, as during flood times, can move boulders weighing 100 tons and more.

The evolution of a landscape following a general uplift above sea level follows definite patterns depending essentially on the amount of rainfall received. In districts with humid climates most rivers flow into the oceans, although interrupted in places by lakes. Four normal stages of evolution of drainage systems in humid areas can be recognized – initial, youthful, mature and old age. During the initial stage the gradients are low, depressions become filled with water, and streams lazily follow various gulleys and flaws in the rock. The rate of erosion is minimal, and few if any placers are formed. As the youthful stage sets in an integrated pattern of major streams with numerous tributaries develops by downward and headward up-gradient erosion, most lakes disappear and V-shaped valleys, and in places steep gulches, canyons and gorges, mark the landscape. With time the normal evolutionary pattern develops further to maturity. Gulch, creek, stream and some river placers are characteristically developed during these stages. As the rivers erode down toward their base level, lateral erosion develops wider and wider valleys with gently sloping sides, flood plains are formed, and meandering courses that wander hither and thither from channel to channel are characteristic. Some creek and stream placers continue to develop, but river, flood plain and deltaic placers are the mark of this stage. Continued slow downcutting of rivers and streams or uplift may leave the remnants of earlier valley floors as terraces in which remnants of the river flood plain placers are preserved. In old age the level of the whole land is reduced to a peneplain characterized by a low relief with low rounded hills, shallow valleys, rolling landscapes and sluggish meandering rivers, marked by oxbow lakes that flow in various directions following the very low gradients that prevail. Few if any placers are formed during this stage.

[33]In heavy mineral studies, mineral grains deposited at the same time are considered as hydraulically equivalent or equi-settling, under the conditions prevailing during deposition. Various factors, principally specific gravity and grain shape, are of importance in determining the relative size of hydraulically equivalent grains. The theoretical and practical aspects of the hydraulic equivalence of grains, with a bibliography, are discussed by McIntyre (J. Geol., v. 67, p. 278–301, 1959).

In arid climates the evolutionary pattern is significantly ifferent, mainly because of reduced rainfall, the absence of ..ck soil cover and the lack of vegetation. Wind erosion is more important in such climates; extensive downhill creep along valley sides is reduced because of the lack of moisture, and numerous basins of interior drainage develop generally with no connections to oceans. The infrequent precipitation during the youthful stage gives rise to numerous nonintegrated streams in steep-walled valleys that cut deeply back into the highlands. With increasing maturity a poorly integrated drainage develops mostly into closed basins that are marked by pediments on which alluvial fans may form. In old age the drainage pattern is generally completely disintegrated, there being in most districts no streams of any length and commonly no interconnection between those that do exist during a pluvial period. Arid conditions are not particularly favourable for the formation of placers, although some may form in gulches, in canyons and in alluvial fans on pediments where the conditions are right. Arid areas are, however, often characterized by great spates and walls of water that come off the highlands or mountains during cloud bursts, rush down the dry washes, gulches, canyons and valleys carrying everything before them, and finally debouch onto the alluvial fans on pediments or into the closed basins. Such violent activity is not conducive to the formation of placers; on the contrary any that formed would be sluiced down the gulches, creeks and canyons and the gold spread indiscriminately over the lower valley bottoms and inland basins.

Desert placers as a whole are small and the pay streaks re often erratic and commonly scattered. 'Bajada' or 'pedi-ent placers' in desert terrains may be more productive, but they are commonly difficult to work. Descriptions of some typical desert placer areas in the southwestern United States and Mexico may be found in the publications by Haley (1923), Webber (1935), Vanderburg (1936), Wilson and Fansett (1961) and Johnson (1972*a,b*, 1973*a,b*, 1974).

The above descriptions characterize the normal evolution of drainage patterns and the formation of placers under the extremes of humid and arid conditions. But many abnormal conditions prevail in any particular district. Sudden uplifts at the mature or old age stages cause renewed downcutting of channels and destruction or deep burial of placers already formed. Similarly, great or sudden increases in rainfall may produce floods that sluice out the gulches, creeks and canyons, change the courses of the rivers, and greatly modify, destroy or bury the most recent placers. Depressions of valleys may result in basins in which lakes develop or inlets are formed, which are invaded by the ocean. These ultimately become filled with sediment burying deeply any placers formed in the valleys. Finally, glaciers forming in high mountains and extending down gulches and valleys may plough out all sediments and scatter the placer materials far and wide.

The location of the pay streaks in placers is of prime interest to the placer miner. Given an adequate primary gold source pay streaks in general are fairly uniform and have considerable continuity in moderately hilly country where uniform rainfall has prevailed and where deep secular decay has been followed by a gradual restricted uplift. Any aberrations in this ideal pattern invariably cause marked variations in the tenor and continuity of the pay streaks. The richest pay streaks are those produced by reworking of preexisting auriferous gravels.

The law of the pay streak in placer deposits is variable depending on whether the placer is formed in gulches, in river channels, on flood plains or in deltaic deposits. Lindgren (1911, p. 66) said of the pay streak referring to the deposits of the Sierra Nevada of California, some of which are buried beneath Tertiary lava flows:

> It has become almost an axiom among miners that the gold is concentrated on the bedrock and all efforts in placer mining are generally directed toward finding the bedrock in order to pursue mining operations there. It is well known to all drift miners, however, that the gold is not equally distributed on the bedrock in the channels. The richest part forms a streak of irregular width referred to in the English colonies as the "run of gold" and in the United States as the "pay streak" or "pay lead". This does not always occupy the deepest depression in the channel and sometimes winds irregularly from one side to the other. It often happens that the values rapidly diminish at the outside of the pay lead, but again the transition to poorer gravel may be gradual. An exact explanation of the eccentricities of the pay lead may be very difficult to furnish. Its course depends evidently on the prevailing conditions as to velocity of current and quantity of material at the time of concentration.

Tyrrell (1912) while admitting the fickle nature of the pay streak concluded that "the pay-streak is a feature in the structure and growth of the valley in which it occurs, its formation is governed by certain geological laws, and those laws should be recognizable without great difficulty if the growth of the valley can be traced with reasonable accuracy."

The features of the evolution of valleys is discussed above, but this knowledge is only of general value in studying placers. To accurately determine the location of their pay streaks, especially in the buried types and those on flood plains and deltas, requires detailed profiling by trenching or overburden drilling. Geophysical methods may assist in some terranes. Where the cover is light and the bedrock magnetically low, a magnetometer survey may outline the position of the magnetic 'black sands' so commonly accumulated with gold. Where the bedrock has a higher magnetic intensity than the gravel, the position of magnetic lows may indicate the position of the pay streak channels. Electrical methods may also assist where there are conductivity or other differences between the bedrock and the basal alluvium. Hammer seismic methods under favourable conditions can also be used for profiling and outlining the notches or gutters containing pay streaks. The pay streak in gulches and youthful streams is generally easily located. It is commonly in or near the notch or gutter of the V-shaped valley or gulch on and just above bedrock (Fig. 71).

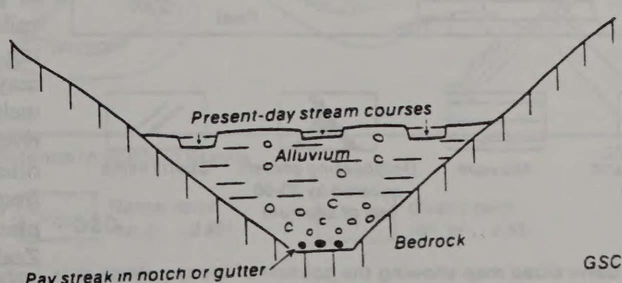


Figure 71. Sketch showing the location of an auriferous pay streak in the notch or gutter of a V-shaped valley or gulch.

As the valleys mature, widen out and become filled with sediments the streams shift their location from time to time, and the pay streaks may no longer be in the lowest depression or gutter. Furthermore, pay streaks long since buried may bear no relation to the present streams (Fig. 72). Pay streaks in wide valleys, flood plains and deltas are extremely irregular in their distribution and are the product of an environment where shifting channels and migrating meanders hold sway (Fig. 73). In most streams and rivers where placers are forming, gold collects along bars due to some obstruction, diversion or slackening of the water course in the slower moving water on the inside curves of fast-flowing meanders, in the main stream near the mouths of tributaries and elsewhere where slack water prevails (Fig. 74). These guides are useful in the study of both recent and ancient water courses. With respect to the latter, however, care must be taken to ascertain the direction of the paleogradients. For instance the Tertiary channels of the Sierra Nevada have been tilted by movements initiated by block faulting so that some of the gold channels now 'run up hill' as compared with the present gradient of the country.

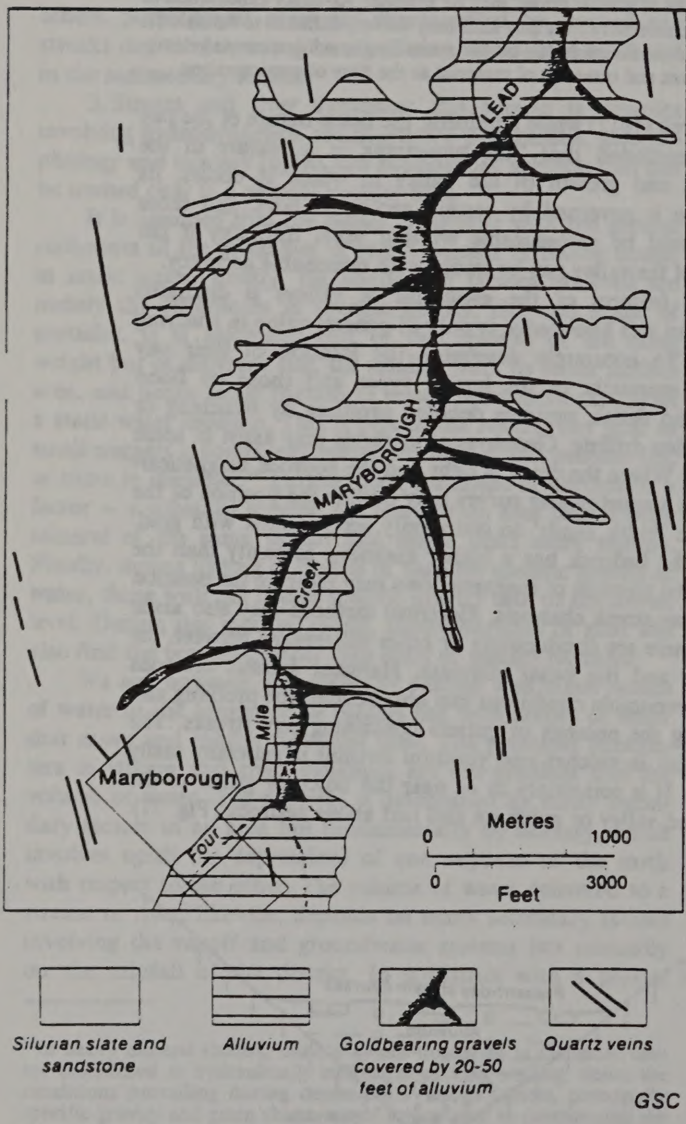


Figure 72. Generalized map showing the position of the pay streak in the auriferous gravels in relation to the present stream system, Maryborough, Victoria, Australia (after Hunter, 1909).

Pay streaks may be reworked by subsequent streams in the same or different courses into exceedingly rich concentrations (Fig. 75). In a broader areal sense the reworking of former placers may also be economically important in places. Thus, the lean placer concentrations of gold and platinum formed during the Mesozoic and early Tertiary in the Urals, U.S.S.R. were uplifted and crossed by a Pliocene-Quaternary drainage system in which rich new placers were concentrated (Sigov *et al.*, 1972).

Deltas may form in rivers, lakes, seas and oceans, but few gold placers in deltas in these locations have been described, probably because they are relatively rare. The principles governing the pay streaks in deltas are the least understood. Deltas in rivers tend to concentrate gold in fairly well-defined streaks just downstream from the mouths of the streams, mainly in topset and foreset beds if these are developed. Deltas in lakes commonly receive only fine-grained to very fine-grained (flour) gold, and the metal may be dispersed more or less throughout the sedimentary pile. In some cases, however, the fine-grained gold may collect in streaks on the foreset beds, and these may have relatively high angles of inclination. Deltas in seas and oceans do not seem to develop well-defined streaks, probably because of the extremely fine subdivision of most of the gold reaching the oceanic environment. If streaks do develop, they are usually erratic and seem to be mainly located in the topset and foreset beds.

An unusual type of (alluvial) placer is described by Bensusan (1942) in Burma. This placer, near Tenasserim, consists of gold in mangrove swamp mud. The gold appears to be present in a colloidal form since repeated examinations of samples and concentrates under a high power microscope failed to show any free gold. From the description it would seem that this is a case of concentration of gold in a humic environment, perhaps by chelation or other organic bonding (*see* also Chapter II). The swamp mud averages about 0.285 ppm Au and is said to contain some £200 million worth of the metal.

At this point a few words should be said about alluvial placers originating from the reworking of glacial debris by glacial and later streams. Some examples of these are given later in the descriptions that follow, and hence only generalities need concern us here.

Valley glaciers give rise to vast terminal moraines and during wasting may clog the valleys with great accumulations of glacial till, clay and sand. If these glaciers have scoured valleys containing gold placers or the sides of valleys containing oxidized or unoxidized gold deposits, the glacial materials may be slightly auriferous. Reworking of these materials by meltwaters and subsequent streams may produce stream and river placers in the normal manner. Many of these placers are often buried, difficult to recognize, and their pay streaks are frequently erratic. Nevertheless, a number of these types of placers have been worked in the past particularly in New Zealand (Park, 1969; Williams, 1974), and there are others in the Rocky Mountains of Canada and the United States, and elsewhere.

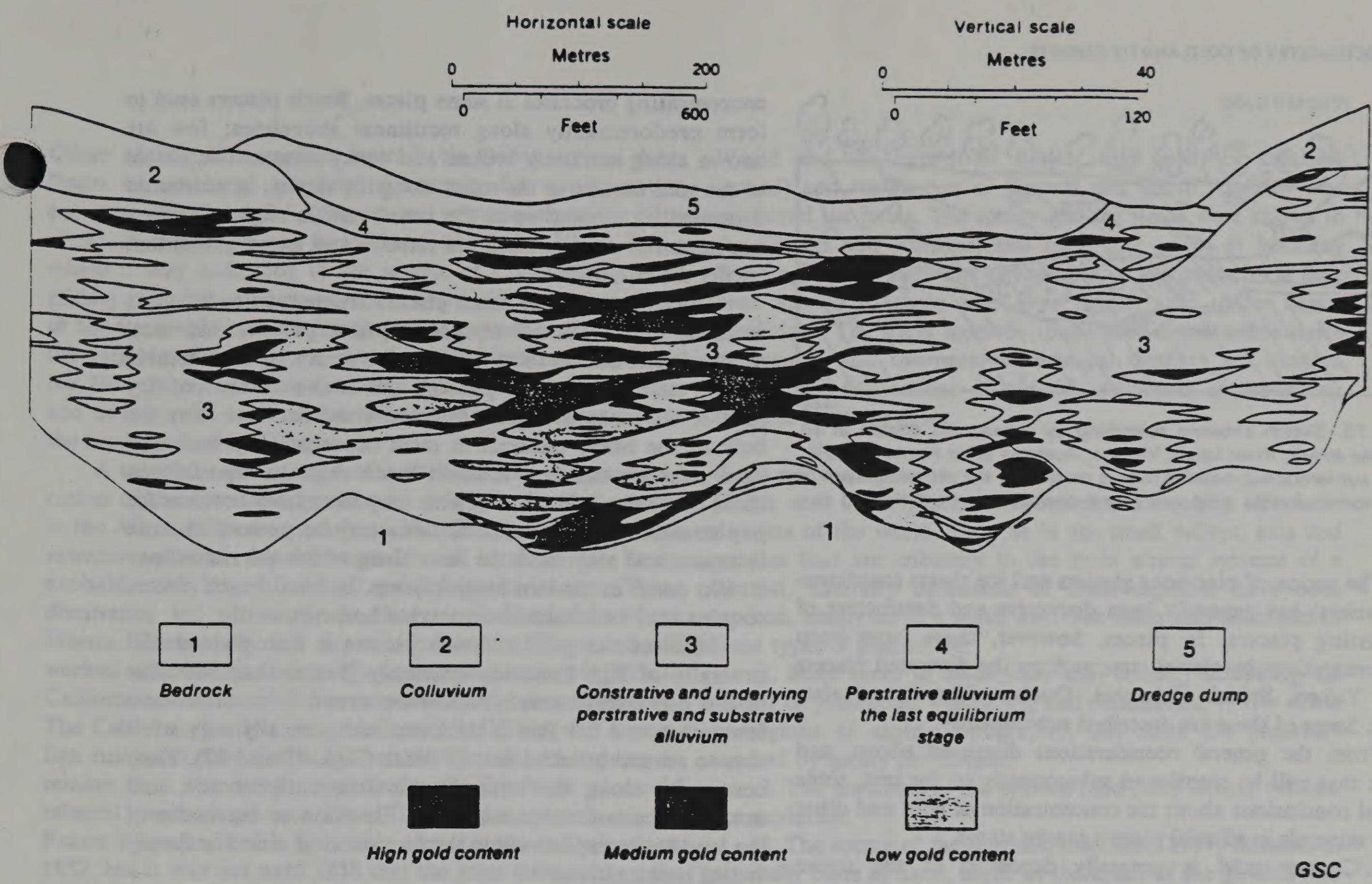


Figure 73. Cross-section of an allochthonous gold placer in the valley of the Kuranakh River, southern Yakutia, U.S.S.R. (after Kartashov, 1971).

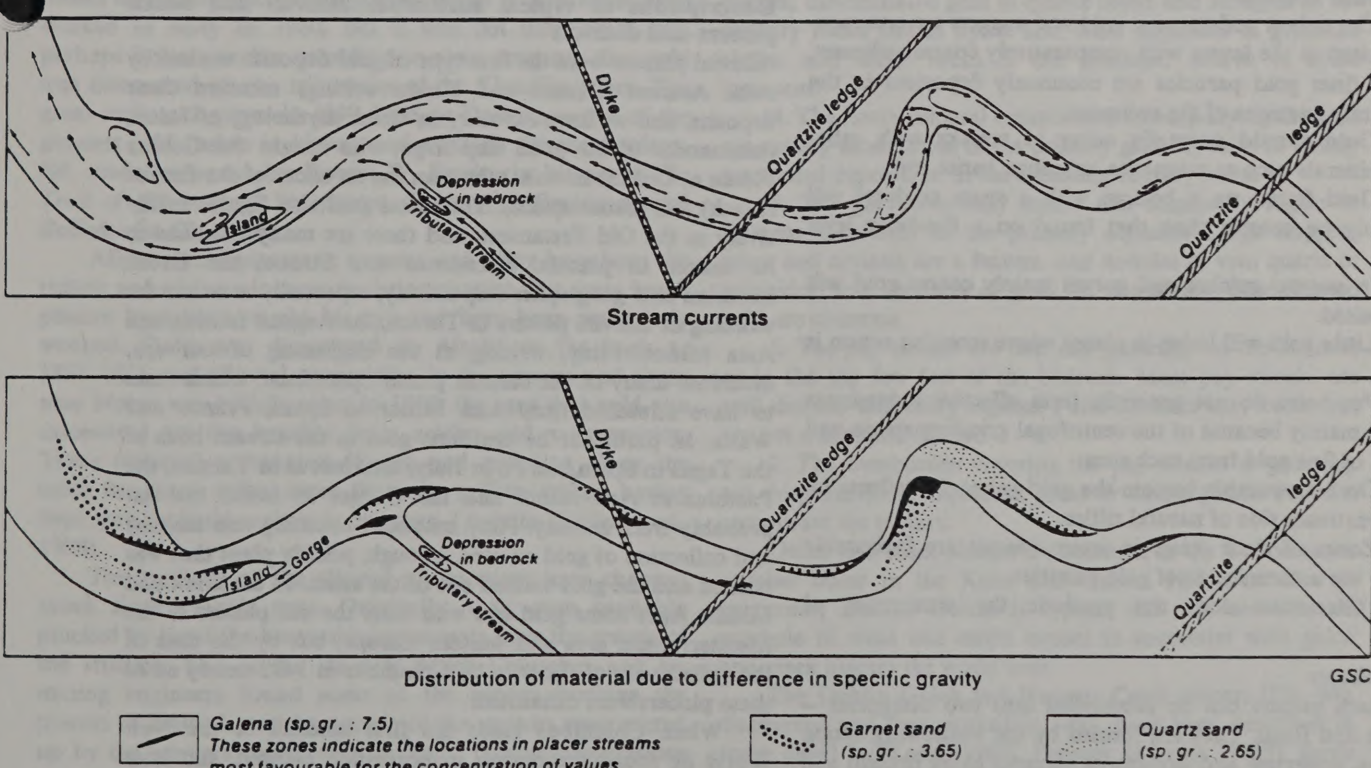


Figure 74. Diagram of a laboratory stream showing currents and distribution of sedimentary materials of different specific gravity (after MacKay, 1921). Galena was employed as the mineral of high specific gravity. The analogy with the placer conditions found in most rivers and streams with moderate gradient is excellent.

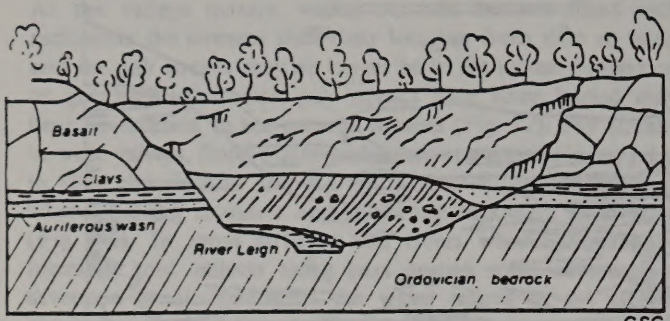


Figure 75. Sketch showing reworking by subsequent stream of an older pay streak, River Leigh, Victoria, Australia (after Hunter, 1909). An old auriferous sub-basaltic river is crossed by the present stream in which some payable gold was found downstream from the old pay streak.

The action of piedmont glaciers and ice sheets (continental glaciers) has generally been disruptive and destructive of preexisting placers. In places, however, there have been reconcentrations by glacial streams from the disrupted placers as in Yukon, British Columbia, Quebec, Siberia and elsewhere. Some of these are described subsequently.

From the general considerations discussed above, and others that will be mentioned subsequently in the text, some general conclusions about the concentration of gold and other heavy minerals in alluvial placers can be stated.

1. Coarser gold is generally deposited in the upper reaches of a placer gulch, stream or river, and the finer grained gold is generally deposited in the lower reaches of placer water courses.

2. The richest and coarsest gold and nuggets are commonly deposited in the layers with comparatively coarse sediment, and the finer gold particles are commonly deposited in the finer, sandy fractions of the sediment.

3. Gold should normally occur in streaks with other heavy minerals such as magnetite, scheelite, barite, etc.

4. Gold found on a bottom with a steep gradient will commonly be coarser than that found on a flat-lying level bottom.

5. In narrow gulches and gorges mainly coarse gold will be deposited.

6. Little gold will lodge in places where scouring action is marked.

7. Pot holes do not generally form effective lodgements for gold mainly because of the centrifugal grinding action and ejection of fine gold from such sites.

8. On a favourable bottom the gold generally collects on the downstream side of natural riffles.

9. Zones of slack water in water courses are favourable sites for the accumulation of gold particles.

10. Glaciation does not preclude the occurrence of placers.

Beach placers

Beach placers can be subdivided into two categories – modern and fossil. They are formed by the winnowing action of waves, undertow and alongshore currents along present and past shorelines bordering lakes, seas and oceans where a source of primary gold has been available for concentration. Tidal movements and strong winds may accentuate the concentrating processes in some places. Beach placers tend to form predominantly along rectilinear shorelines; few are known along markedly broken and rocky shores. The source of the gold may be in the rocks along the shores; in sediments deposited by streams upon the coast; and in slightly auriferous stream or river terraces, sea terraces and gravel plains that border the coasts.

The constituents of beach placers are essentially the same as those in the alluvial category. Quartz pebbles and sand predominate, but on some beaches there may be considerable shingle composed of the pebbles and stones of the country rock of the adjacent region. Magnetite and ilmenite form the bulk of the heavy minerals in most deposits. Clay beds and hard pans are common in some beach deposits especially in those of a raised character. These may form false bottoms for pay streaks. Coquina accumulations may be present in some instances and may mark the lines along which the richest pay streaks occur in modern beach placers. In fossil beach placers coquina may be invaluable as marker horizons.

Most of the gold in beach placers is fine grained and generally of high fineness, commonly greater than 900. The richest pay streaks usually follow the strand lines; most are a few inches to a few feet in thickness and generally only a few tens to several hundred feet in width (Figs. 88 and 89). Their continuity along the strandline is frequently erratic, and numerous cutouts are encountered. Elevation or depression of the land leads respectively to the formation of raised or deeply buried beach placers.

Descriptions of typical auriferous alluvial and beach placers and districts

Alluvial placers were the first type of gold deposits worked by man. Ancient Chinese and Hindu writings mention these deposits, and we have the story in Greek mythology of Jason, commander of the good ship *Argo*, who sought the Golden Fleece at Colchis in Asia Minor on the far shore of the Euxine (Black) Sea (Frontispiece). The metal gold also figures extensively in the Old Testament, and there are many allusions to its source in placers. Herodotus and Strabo, the Greek historian and geographer respectively, repeatedly mention the working of alluvial placers in Thrace, the Aegean Islands and Asia Minor. Pliny, writing at the beginning of our era, describes many of the Roman placer operations, which seem to have stretched from Asia Minor to Spain, France and Wales. In particular he mentions gold in the stream beds of the Tagus in Spain, the Po in Italy, the Hebrus in Thracia, the Pactolus in Asia Minor and the Ganges in India, the last probably from heresay. Pliny mentions booming and sluicing and collection of gold on ulex, a rough, prickly plant that was burned and the gold washed out of the ashes. In the Dark and Middle Ages some gold was won from the old placers of the Mediterranean area and western Europe, but by the time of the rediscovery of America by Columbus in 1492 nearly all of these placers were exhausted.

When Columbus made his first landfall in the New World he found the natives in possession of gold, and later exploring in Hispaniola (Haiti) in 1493 on the second voyage he observed crude placer operations in the streams and rivers of the interior. In 1494 Alonso de Ojeda discovered the rich

Cibao placers, and a year later Pablo Belvis arrived from Spain with a large quantity of mercury for amalgamation purposes. The first gold won was sent immediately to the King of Spain, who donated it to Pope Alexander VI in Rome where it was dedicated to the service of Christianity in the gilding of a cathedral dome. Thus began the *auri sacra fames* of the Spaniards in America that was to wreak such havoc on the natives and was also to introduce the Negro slave trade to the New World. The sordid story has been told many times and need not be repeated here; it is one of the black pages in the history of gold mining.

A more pleasant story can be written about the great gold rushes of the last century. Alluvial gold gravels were worked in the Altai of Siberia as early as 1820, but some 9 years later extensive alluvial deposits were found in the Lena Basin, probably the largest alluvial gold deposits known. These discoveries led ultimately to the colonization of southern Siberia. These and other deposits in eastern Siberia are still worked extensively today. The great gold rush of 1849 to California can be said to have opened up the American west. The California gold rush was followed in 1851 by the Australian rushes to New South Wales and Victoria, an impetus to mining that has kept Australia in the forefront of world mineral production ever since. The golden gravels of the Fraser River in British Columbia were known as far back as 1852, but it was not until 1858 that the great stampede up the Fraser began ultimately in 1861 reaching Williams and Lightning creeks in the Cariboo, the most celebrated of all gold creeks in the province. Gold placers in the Yukon Basin were worked as early as 1880, but it was not until 1896, and perhaps 2 years earlier according to some accounts, that gold was discovered on the tributaries of the Klondike River. The great rush took place in 1897–1898 to Dawson – one of the greatest gold rushes in history and certainly the most colourful, made immortal by Robert W. Service in his novel *The Trail of '98*, and in his poems, *Songs of a Sourdough* and *Ballads of a Cheechako*.

Alluvial gold placers in streams and rivers have been the richest and of most interest to the prospector. But some beach placers have also caught his eye and have been extensively worked. Gold was discovered in Alaska as far back as 1865–1866 and alluvial prospecting in the streams and rivers near Nome was in full swing in 1899, the year that gold was discovered on the beaches by a soldier and a prospector. There followed a frenzied digging and grubbing along the coast for many miles, more than one million dollars having been won by hand rockers in less than 2 months (Collier *et al.*, 1908).

The methods of the alluvial placer miner have changed much since Jason's time. Originally, early man probably plucked by hand the heavy shining nuggets from the gravels of the streams. This is the stage at which Columbus and his mining engineers found some of the natives working the placers of Haiti. Later the gravels of the streams were stirred up by the workers using crude booming (hushing) operations and sluiced over the fleeces of sheep and goat pelts, the gold remaining mainly trapped within the wool and goat's hair. This method is apparently still employed in some of the placer streams of Asia Minor, Afghanistan and Mongolia. Panning is an old technique certainly known to the ancients of the Old World and the natives of Africa, who used the calabash (gourd) and the natives of Central and South America who employed the batea. The rocker and the sluice were known to the Greeks and Romans, and they were adept at booming (hushing) operations. The dry washer or dry blower has long been known to those who have sought gold under desert conditions. The water monitor-, bulldozer- and dragline-sluice operations and the great mechanical dredges are modern mechanical adaptations of age-old techniques of separating the gold from the dross.

Gulch and creek placers

Gulch and creek placers are common in nearly all placer districts of the world. They lie in the small valleys, guts and gulches that are tributary to the main stream systems of a district. Literally thousands of these deposits have been worked, nearly all in a small way. The main characteristics of these types of placers are:

1. Most occur in moderately hilly country exhibiting the effects of protracted weathering and denudation. A few occur in regions of alpine topography, but these are generally marked by spotty pay streaks.

2. The gradients of the present (and past) stream systems are moderate.

3. The source of the gold and associated heavy minerals is generally close at hand, either at the heads of the gulches and creeks or along their valley sides. The source of the gold is commonly auriferous quartz veins or gold-bearing sulphide bodies; disseminated gold in quartz blows and stringers in the country rocks and in pyrite and other sulphides in graphitic shales and other rocks is the principal source in some instances.

4. The heavy mineral constituents accompanying the gold are mainly those found in the primary gold deposits, in closely associated deposits or in the enclosing country rocks.

5. The gold is usually coarse and commonly higher in fineness than that in the primary deposits. Large nuggets, wires and crystals are a feature, and nodules of vein quartz or sulphides with veinlets of gold or containing disseminated gold are common.

6. The pay streaks are rich and generally on the bedrock or in the top few feet of the bedrock. Most pay streaks are well defined and fairly regular. False bottoms may occur but are not a common feature.

7. The overburden covering the pay streaks is generally not deep except in depressed areas or where glacial deposits complicate the picture.

Gulch and creek placers are common in the Yukon and a number occur in the Keno Hill–Galena Hill area. One in particular, Dublin Gulch, is typical and will serve as an example of what one might expect to encounter with gulch and creek placers the world over.

The Dublin Gulch and Haggart Creek placers (Fig. 69), discovered and first worked in 1898, have been described in some detail by Keele (1905), Bostock *in* Little (1959), Boyle (1965*a*) and Boyle and Gleeson (1972). The topography is characterized by relatively low rounded hills and numerous small streams tributary to the main water courses, all marks of a deeply dissected upland. The bedrock of the area is folded and faulted quartzite, phyllite, graphitic schist, limestone and

quartz-mica schist of the Yukon Group intruded by a body of granodiorite and granite probably of Cretaceous age. Skarn, carrying scheelite, is developed in places near the granitic rocks. In addition there are numerous quartz stringers carrying wolframite and scheelite in the granitic rocks and their contact zone, a cassiterite-tourmaline lode and several northeast striking gold-bearing quartz-arsenopyrite-pyrite-sulphosalt veins. Primary vein material in these veins averages about 0.2 oz Au/ton and 1.5 oz Ag/ton. The gold is largely submicroscopic to microscopic and is mainly in the arsenopyrite, pyrite and sulphosalts; only rarely is free gold seen in the quartz and sulphides.

The country rocks and the gold-bearing veins are deeply weathered, the latter down to at least 20 ft. The stable minerals in the oxidized zones of the veins are mainly scorodite, limonite, residuals of pyrite and arsenopyrite and some anglesite and beudantite. Assays of the oxidized material are highly variable in gold content, ranging from 0.015 to 0.30 oz Au/ton; the silver content is, likewise, highly variable averaging about 0.5 oz/ton. The gold in the oxidized zones consists of fine dust, small flakes and abundant wires. Some of the gold is closely associated with the scorodite and limonite and is probably in a chemically bound or adsorbed form. Most of the gold in the oxidized zones is about 850 fine, although there are marked differences from vein to vein.

The area immediately around Dublin Gulch is beyond the limit of the last well-marked Pleistocene glaciation, but evidence of an early glacial incursion is present as witnessed by what seem to be remnants of weathered tills and also by the appearance in the placer concentrates of nodules of hematite (iron-formation), which probably came from the Snake River deposit in northern Mackenzie Mountains, Yukon. The streams in Dublin Gulch and Haggart Creek have entrenched parts of their courses in deep overburden forming terraces with old modified profiles. The exact nature of the original profile in the lower part of Dublin Gulch is difficult to ascertain because of the extensive stirring and digging since 1898. What appears to have been present at the surface was about 6 to 8 ft of gravels, schist particles, soil and granite boulders, probably mainly due to creep and slope wash. This was succeeded in places by 2 to 3 ft of bluish clay that overlay 3 to 4 ft of yellowish (limonite coated) gravels and weathered debris. The last materials represent the detritus arising from deep secular weathering of the bedrock of the area and the contained deposits during a period of uplift in late Tertiary time.

According to old reports fine colours of gold occurred in the surface materials where washed by the stream in the gulch. Most of the gold, however, is on the bedrock in the bottom foot or so of the lower gravels and weathered debris. The pay streak begins at Bawn Bay Gulch and continues with only a few breaks downstream. It is about 100 ft wide near the mouth of Dublin Gulch, narrowing gradually upstream. The grade is about 0.03 oz/yd.

The gold is accompanied by scheelite (about 1 lb/yd^3), some wolframite and a little cassiterite. In addition there are a number of other heavy minerals, including magnetite, hematite nodules (iron-formation), arsenopyrite nodules, jamesonite nodules, bismuth, galenobismutite, rarely bismuth tellurides, pyrite, tourmaline and garnet. The gold is present as fine dust, scales, rough wires and sprigs, occasional crystals and small nuggets ranging from pea size up to about the size of hickory nuts. Most of the nuggets are worn and pitted. The writer did not observe any free gold in the quartz or sulphide pebbles and so far as is known none of this type of gold has been reported. Mustart (1965), however, found angular arsenopyrite fragments in some of the nuggets. The principal minor and trace metals in the gold are Ag, Cu, Fe, Hg, Bi and Sn. All of these occur in the deposits from which the gold came. The average fineness of the Dublin Gulch gold is about 900.

The Dublin Gulch area provides an excellent example where gold can be traced from primary deposit to stream placer. The primary gold is mainly microscopic to submicroscopic and associated with pyrite, arsenopyrite and sulphosalts. Particles of gold greater than 100 μ are rare in the writer's experience. On the oxidation of the sulphides and sulphosalts the gold was released and formed flour or mustard gold in some cases, but there was also a nucleation and growth of gold on fine rough wires, flake and small nuggets in the oxidized zones. All of these types of gold passed gradually into the eluvium where they probably accreted more gold on their journey to the gulch. Here, they passed slowly down gradient, some particles possibly accreting more gold but most being rolled along and hammered by the pebbles of the mills of nature. The fineness has changed during the journey from oxidized zone to placers in Haggart Creek a distance of some 3 mi. In the oxidized zones the fineness averages about 890; in Dublin Gulch 900 and in Haggart Creek 925.

Stream and river placers

The main features of these placers are discussed in some detail above. Briefly their characteristics are: (1) most occur in districts with a subdued topography marked by broad, often terraced, entrenched valleys developed in a terrain of rounded, deeply weathered hills; (2) the present (and past) gradients of the streams are moderate to low; (3) the pay streaks are generally not as rich as those in gulches and creeks, but they are longer, commonly wider and more uniform. Multiple pay streaks may occur at the same elevation or they may be stacked at different elevations. Most of the rich pay streaks are on the bedrock; (4) the gold is generally finer grained than that in creek and gulch gravels. Large nuggets are relatively rare in stream and river placers; (5) the fineness of the gold is usually higher than that in creek, gulch and eluvial placers; (6) the overburden is commonly tens of feet to hundreds of feet deep; and (7) most stream and river placers are amenable to large scale hydraulicking and dredging operations.

There are many famous districts where stream and river placers have been extensively worked. Some provide good examples and will be described briefly. They include the Klondike district in Yukon, the Cariboo and other districts in British Columbia, the Chaudière and other areas in southeastern Quebec, the Sierra Nevada of California; the Chocó in Columbia, the deep leads of Victoria, Australia, and the extensive placers of the Lena, Aldan, Amur and other drainage systems in Siberia, U.S.S.R., Mongolia and northern China.

Klondike district, Yukon. The stream and river placers of the Klondike district near Dawson, Yukon were discovered in 1896 or earlier and have yielded some 10 million ounces, more

UNITED STATES DEPARTMENT OF THE INTERIOR

GEOLOGICAL SURVEY

BIOGEOCHEMICAL STUDIES OF GOLD IN A PLACER DEPOSIT, LIVENGOOD, ALASKA

By

R.J. Coel,[1] J.G. Crock,[2] and J.R. Kyle[1]

Open-File Report 91-142

This report is preliminary and has not been reviewed for conformity with U.S. Geological Survey editorial standards and stratigraphic nomenclature. Any use of trade names is for descriptive purposes only and does not imply endorsement by the USGS.

[1]Department of Geological Sciences, University of Texas at Austin, Austin, TX 78712
[2]U.S. Geological Survey, DFC, Box 25046, MS 973, Denver, CO 80225

1991

CONTENTS

ILLUSTRATIONS

TABLES

BIOGEOCHEMICAL STUDIES OF GOLD IN A PLACER DEPOSIT, LIVENGOOD, ALASKA

By

R.J. Coel, J.G. Crock, and J.R. Kyle

INTRODUCTION

Gold is generally regarded as chemically inert at the earth's surface, and the movement of gold within sediments and soils has long been considered a mechanical process. However, there is evidence for the presence of mobile forms of gold in the surficial environment. Gold has been reported in the parts per billion (ppb) range in plants (Shacklette and others, 1970; Severson and others, 1986; Cohen and others, 1987; Jones and Peterson, 1989) and in the parts per trillion (ppt) range in surface waters (Gosling and others, 1971; McHugh, 1984), indicating that gold is chemically active under ambient temperature and neutral pH conditions (Lakin and others, 1974).

Concentrations of gold are most commonly reported as total gold in soils and sediments, and there is little quantitative information regarding the chemical forms of gold. This information is difficult to obtain because of the generally low concentration of gold in these materials. A sensitive analytical method is necessary to determine gold in a certain fraction of a sample, for example, gold associated with organic material. Partial, sequential extractions are commonly used to determine the mode of occurrence of transition metals (Chao, 1984). Few such extractions are useful for determining gold's modes of occurrence, however, due to the unique properties of gold such as its ionic instability in most solutions. Therefore, controversy still exists over the most common forms of gold, other than elemental, in the surficial environment.

An understanding of gold's geochemistry and mobility in the weathering zone is important to the interpretation of geochemical and biogeochemical surveys. The placement and extent of a geochemical or biogeochemical gold anomaly is strongly influenced by the mobility of gold in a given environment. Knowledge of the chemical behavior of gold at ambient temperatures should be considered essential to the design and evaluation of an exploration survey.

One problem that has plagued both gold exploration and analysis is the "nugget effect." This is the tendency of gold to be present as a few particles in a large amount of matrix. The nugget effect may be minimized by taking very large samples in order to accurately represent the sampled population. By sampling plants or decayed plant material in which gold would not be expected to occur as nuggets, it may be possible to identify gold anomalies more accurately. However, the implementation of a biogeochemical survey requires an extensive knowledge of the factors that influence the gold content of a plant and its relationship to the gold content of the substrate.

A placer gold mining operation in the Tolovana gold district of central Alaska was chosen as a study area to investigate the behavior of gold in the surficial environment. In a preliminary study by Severson and others (1986) on Lillian Creek, several species of plants in this area including labrador tea, alder, and birch, were found to contain anomalously high levels of gold. In the same study, water-extracts of soil samples yielded measurable amounts of gold in the low ppb range. This work suggested that the Lillian Creek gold placer deposit would be an appropriate study area to investigate the mobility of gold in the weathering environment.

Soil and plant samples were collected during the summer of 1988. The samples were analyzed for a variety of chemical constituents in order to determine the distribution, and to some extent, mode of occurrence of gold in natural materials at the placer. Gold in plants and water-extracts of soils are used as a measure of gold activity (Lakin and others, 1974). The abundance and distribution of potential gold-mobilizing agents in the soils, as well as other elements associated with gold in these types of deposits, such as arsenic and antimony, were also investigated. In addition to the analysis of samples from the study site, controlled dissolution experiments were performed in the laboratory to investigate gold's behavior in the surface environment.

The objectives of this study were: (1) to describe the distribution of gold in the soil and two species of native plants; (2) to confirm the existence of mobile forms of gold in the surficial material; and, (3) to suggest probable agents of gold mobilization using the results of both observational and experimental studies.

GOLD GEOCHEMISTRY AND BIOGEOCHEMISTRY

Gold and Inorganic Ligands

Observing gold in soil solutions raises the question of gold mobility in the weathering zone. Due to their high oxidation potentials, the simple ionic forms of gold, Au(I) (+1.68 volts) and Au(III) (+1.50 volts), are not stable in water (Ong and Swanson, 1969). Therefore, gold must be transported as a complex ion or ion pair. To form a complex, gold must first be oxidized to a simple ion, and the strength of the oxidizing agent necessary is dependent upon the stability of gold complex being formed. Lakin and others (1974) have reviewed the stable complexes of gold at 25 °C with a number of inorganic ligands such as chloride, bromide, iodide, cyanide, thiocyanate, and thiosulfate.

Gold is a type B metal ion ("soft" acid) and is therefore more likely to bind to a type B ligand ("soft" base) according to the general rules by Pearson (1973). Although these rules are largely empirical, there is a theoretical basis for the observation: hard acids and bases have a tendency to form bonds that are more ionic in nature, whereas soft acids and bases generally form covalent bonds (Huheey, 1983). Arranged in the order of increasing softness and stability of gold complexes formed, some potential, inorganic gold complexing ligands include (from Renders and Seward, 1989):

$Cl^- < Br^- < SCN^- < I^- < S_2O_3^{2-} < CN^-$

increasing stability--------->

hard<-------------->soft

At ambient temperatures, atmospheric oxygen is an adequate oxidizing agent for the formation of gold cyanide (Lakin and others, 1974), but a stronger oxidizing agent, such as manganese dioxide, is required for the formation of gold chloride (Krauskopf, 1951). Lakin and others (1974) reviewed these inorganic ligands and stated that in an oxidizing environment where pH values range from 5 to 8, only thiosulfate, thiocyanate, and cyanide ions are likely to complex with gold.

Gold chloride complexes are cited by Helgeson and Garrels (1968) as the most important inorganic gold-complexing agent in natural aqueous solutions. However, the solubility of gold in the presence of chloride drops sharply with decreasing temperature (Henley, 1973). In view of this temperature dependency and the requirement of a strong oxidizing agent to form gold chloride complexes, except under unusual conditions, these complexes can be expected to play a limited role in gold mobility in the surficial environment. Bromide and iodide complexes of gold are more stable than chloride complexes (Lakin and others, 1974), but bromine and iodine are much less abundant than chloride in soil. According to Vinogradov (1959), the average soil concentration of both bromine and iodine is 5 parts per million (ppm), compared with 100 ppm for chlorine. Base metals such as copper and iron form more stable complexes with bromide and iodide than does gold, and will therefore be competitors for the limited amounts of bromide and iodide present in soils and sediments (Lakin and others, 1974). Except in unusual circumstances, bromide and iodide are unlikely gold complexing agents in the weathering zone.

Several anionic sulfur compounds are potential complexing agents for the transport of gold at 25 °C. Seward (1973, 1982) reported $Au(HS)_2^-$ to be a stable complex at near-neutral pH values under reducing conditions. However, the weathering zone is generally oxidizing and the thiosulfate ion is more likely to complex gold than biosulfide. Webster (1986) stated that thiosulfate would be relatively stable under the mildly oxidizing conditions present in the phreatic zone, whereas sulfate would be the dominant sulfur species in the more oxidized soils above the water table. The tendency of sulfur to oxidize to sulfate in the soil limits the abundance of compounds such as thiosulfate, which contains sulfur in an intermediate oxidation state. The oxidation of sulfides was shown by Goldhaber (1983) to yield thiosulfate at pH values above 7. Under weakly acidic conditions, thiosulfate was either not formed, or it was oxidized to tetrathionate before it could be detected. Lakin and others (1974) cited both a lack of stability and abundance of this compound as the limiting factors in thiosulfate's gold-mobilizing capabilities. However, considering the relative stability of gold thiosulfate compounds, the possibility of gold transport as a mobile complex with thiosulfate should not be discounted, especially in neutral to alkaline soils with high sulfur concentrations.

Cyanide forms very stable complexes with gold and is a likely agent of gold mobilization in the soil environment (Lakin and others, 1974; Warren, 1982). Shacklette and others (1970) showed that gold cyanide complexes are readily absorbed by plants. However, because cyanide

is dependent upon cyanogenic plants and microorganisms for its synthesis, cyanide is not ubiquitous in soils and sediments. The tendency of cyanide to hydrolize also limits its concentration in soils (Lakin and others, 1974).

Thiocyanate has also been found to form stable gold complexes, but it is readily oxidized by the soil bacteria *Thiobacillus thiocyanoxidans* and, probably is also limited in its abundance in surficial environments (Lakin and others, 1974).

Gold and Soil Organic Matter

The affinity of metals for organic material is well known. High concentrations of metals are often found in organic-rich deposits such as coal and carbonaceous shales. The association of gold with organic material has been noted both in hydrothermal gold deposits (Levitskiy and others, 1983) and in the Witwatersrand paleoplacer (Dexter-Dyer Grosovsky, 1983). Gold is known to form a number of complexes with organic compounds (Johnson and Davis, 1973), some of which may have an influence on gold mobility in the soil environment (Boyle, 1979).

Humic substances are often defined as the soil organic matter which cannot be classified as discrete organic compounds such as amino acids, carbohydrates, or lipids (Stevenson, 1982). Humic substances are defined by Schnitzer (1978) as "dark colored, acidic, predominantly aromatic, hydrophilic, chemically complex, polyelectrolyte-like materials that range in molecular weights from a few hundred to several thousand." Humic substances are broken down into three main categories: humic acids, fulvic acids, and humin. These substances are operationally defined based on their relative solubility in dilute acid and base. Humic acids are alkali soluble, but acid insoluble; fulvic acids are both acid and alkali soluble; and humin is both acid and alkali insoluble (Schnitzer, 1978). Many other properties separate fulvic acids from humic acids: fulvic acids are lighter in color, have a lower molecular weight, contain less carbon and more oxygen, and have more exchange sites per unit weight relative to humic acids (Stevenson, 1982). The greater solubility and exchange capacity make fulvic acid the primary focus of the experimental portion of this study.

Humic acids are thought to play an important role in controlling the availability of both nutrients and toxic substances to plants (Schnitzer, 1978; Stevenson, 1982). Humic acids have also been shown to form strong complexes with base metals, such as copper and iron (Van Dijk, 1971). The ability of humic acids to complex or adsorb gold has been postulated and not proven conclusively. Curtin and others (1970) demonstrated that gold concentrations in organic material leachates to be roughly proportional to color; that is, the darker the leachate, the higher the gold content. Because soil organic matter is credited for imparting a dark color to soils and waters, this observation implies an association between gold and organic substances in gold-bearing soils. Baker (1978) was able to measure up to 330 ppb gold in solution after exposing particulate gold to solutions of humic acid. This represents an enrichment of several thousand times the amount of gold usually found in natural waters (McHugh, 1984). Other authors have cited humic and (or) fulvic acids as possible complexing agents for gold (Freise, 1931; Rashid and Leonard, 1973; Baker, 1978; Boyle, 1979; Fedoseyeva and others, 1985; Severson and others, 1986; Jones and Peterson, 1989). The results of other studies have not always supported the ability of humic substances to complex gold (Fetzer, 1934; Andrade and others, 1988). Ong and Swanson (1969) suggested that humic acids do not form chemical complexes with gold but

instead form a protective, hydrophilic coating around gold colloids, allowing the gold to remain in suspension.

Gold in Plants

Although there are many reports of anomalous levels of gold in plants, very little is known about the physiological processes that control the uptake of gold by vegetation. Controlling factors may include the age and growth rate of a plant, the type of plant tissue of the plant, for example, stems versus leaves, and the uptake physiology of the plant (Berry, 1986). The "uptake physiology" refers to the ability of a plant to exclude, accumulate, or passively adsorb a given element, and is dependent both on the species of plant and the element's concentration in the soil. All of these factors should be taken into account before implementing, and especially interpreting, a biogeochemical survey. Many soil factors also influence elemental abundance in plants. Thornton (1986) cited drainage conditions, pH, chelating agents in the soil, antagonistic and synergistic effects of other elements, and the effects of soil microorganisms (which are poorly understood), as soil factors affecting the uptake of metals by plants.

In a comprehensive study of the ability of plants to absorb various forms of gold, Shacklette and others (1970) concluded that impatiens (*Impatiens holstii*) and garden balsam (*Impatiens balsamina*) are unable to assimilate colloidal gold. This study, which measured the movement of radioactive gold (Au^{198}) through these two plants, tested the ability of both rooted and cut plants to assimilate gold as a complex with chloride, cyanide, bromide, iodide, thiosulfate, and thiocyanate, as well as two different sizes of gold colloids (<0.05 µm and 0.05-0.45 µm). Autoradiographs of the leaves of cut and rooted plants showed that radioactive gold from the colloidal mixtures was not transported to the leaves. The plants with intact roots were able to transport radioactive gold to their leaves from all but the gold chloride and thiosulfate solutions. The leaves of plant cuttings were found to absorb Au^{198} from all of the solutions tested. Only the radioactive gold content of the leaves was investigated in this experiment. Therefore, the amount of gold assimilated by other parts of the plants is unknown.

A similar experiment was performed using nonradioactive gold to determine if the plants would preferentially assimilate Au^{197} from the various solutions (Shacklette and others, 1970). Analysis of stems and leaves by atomic absorption spectroscopy showed that gold was absorbed by plants from all of the solutions in varying amounts. These results suggest either that Au^{198} is not as readily absorbed as Au^{197}, which is unlikely, or that certain solutions of gold are sequestered in the stems and are not transported to the leaves in appreciable amounts. However, autoradiographs of Au^{198} may not be as sensitive in detecting gold as atomic absorption spectroscopy. This experiment did not investigate the absorption of nonradioactive colloidal gold by plants, and therefore the ability of plants to absorb small colloids of Au^{197} remains in question.

The uptake of many metals, including gold, by three species of conifers was investigated by King and others (1984). In this study, lodgepole pine, douglas fir, and engelmann spruce seedlings were planted in pots containing soil and characteristic ore minerals of various types of ore deposits. These seedlings, along with a control group planted only in soil, were moved to a coniferous forest in Jefferson County, Colorado. The pots were placed in the ground so that the surface of the soil in the pots was approximately even with the ground surface. The seedlings

were allowed to grow under natural conditions for seven years. They were then removed from the plot and analyzed for various metals. The results showed that many metals from the ore minerals had been assimilated by the plants. Gold, which was supplied to the plants as gold foil, was found in the ppm range in the ash of roots, leaves, and stems of some of the plants. The highest concentrations of gold were generally found in the roots, but the authors speculate that this may be due to contamination by mineral matter on the outer surfaces of the roots. The findings of this study are significant because they show that metallic gold in the soil may be converted to a form that is available to plants in a relatively short time.

FIELD STUDY

Location

The field study area is located within the Lillian Creek drainage (T. 8 N. R. 5 W., sec. 22) which runs east to west approximately 1.5 km south of Livengood, Alaska. Livengood is located approximately 90 km north-northwest of Fairbanks, Alaska (fig. 1a).

Climate and Topography

The study area is located within the Yukon-Tanana uplands in the interior basin of Alaska where the climate is semiarid, subarctic (Mertie, 1937). The mean annual temperature of Fairbanks is -3.4 °C, and ranges from -54.4 °C to 33.9 °C. Annual precipitation in Fairbanks is 28.7 centimeters, most of which falls in the summer months (Hare and Hay, 1974).

The topography of the area is well described by Mertie (1918):

> (p. 222) The Tolovana district is part of the Yukon-Tanana upland. Its topography is characterized by broad, even-topped ridges, which rise to a general elevation of 2,000 feet or higher and from which long, gently sloping spurs descend to the valley floors.
>
> (p. 229) The ridge-tops in the Yukon-Tanana region as a whole are probably the dissected remnants of an old peneplain, in the widest sense of that term...more or less uniform in elevation but sloping shieldlike to the major drainage channels.

The Yukon-Tanana region is located within the zone of discontinuous permafrost (Péwé, 1975). Permafrost areas were noted in the study area, especially on the north sides of the drainages. The south-facing side of the Lillian Creek drainage is less steep than the north-facing side, due to the thawing and solifluction of the unconsolidated material on the south-facing slopes.

Vegetation

Livengood is located within the interior spruce and birch forest region, consisting chiefly of white and black spruce, birch, poplar, and aspen (Sigafoos, 1958). Timberline in the Yukon-Tanana area is 758 m (2,500 ft) above sea level (Mertie, 1937), but trees may be absent to

DISCUSSION

Geochemical and Biogeochemical Anomalies

The analytical results for soil total element concentration are given in tables 5 and 6. The distribution of gold, arsenic, and antimony in the soils supports the presence of bedrock mineralization in the upper portion of the drainage (Mertie, 1918; Foster, 1969; Eakins, 1974). Most of the high values for these elements occur upstream of sample site 4, with the exception of sample 7-20+, which contains high concentrations of all three of these elements. The arsenopyrite-quartz vein which was found to contain 90 ppm gold runs roughly perpendicular to the drainage near sample site 3 (fig. 1) and may account for some of the high Au and As values found in soils there. The highest antimony values are centered around sampling sites 2a and 2b.

The distribution of base metals, particularly chromium, nickle, zinc, copper, and iron, in the soil also support the presence of mineralization in the Tertiary intrusions in the upper reaches of the drainage (sites 1-4), particularly in the vicinity of sites 2a and 2b.

The pattern of gold, arsenic, and antimony concentrations in plants from the Lillian Creek drainage, like the soil anomalies for these same elements, support the presence of bedrock mineralization in the upper part of the drainage. Only one plant sample was found to be a "gold accumulator," that is, the plant is capable of absorbing gold in amounts greater than those present in the substrate (Berry, 1986). The labrador tea collected at site 3 contained greater than an order of magnitude more gold than any of the soil samples collected at that site (3,700 ppb vs 100 ppb). One possible explanation for the extremely high amount of gold found in this plant sample is the presence of the arsenopyrite-stibnite-quartz vein in the vicinity of site 3. Another factor which may have caused the absorption of elevated levels of gold by this plant may be the apparently slow growth rate of the plants at this north-facing sample site (Berry, 1986) or from the unusually high metal content of the soil. This sample site is unlike all of the others in that it is more "tundra-like" in its characteristics. There are few trees, and the vegetation is dominated by low-growing species, such as mosses, lichens, and blueberries. The alders and labrador tea present at this site appear stunted, and it took many individual plants to make up an adequate sample. A slow growth rate may contribute to elevated metal content in plants because a plant may continue to absorb metals over the same period of time, even though it is growing slowly and has a smaller mass (Berry, 1986). Increased ice activity on the north-facing side of the valley results in poor drainage which may also be a factor in determining the metal content of plants. Poorly drained soils contribute to higher levels of metals in plants (Thornton, 1986). Watterson (1985) speculated that mobilization of gold may be increased by the concentration of soil components such as gases, organic acids, and mircoorganisms into the bound water during freezing. Theoretically, this effect may also contribute to the elevated levels of gold in plants, such as that observed in the labrador tea collected at site 3.

The alder collected at this site also contained a relatively high level of gold (3.1 ppb), but was not found to be a gold accumulator. Differences in gold contents between labrador tea and alder are probably accountable for by differences in uptake physiology.

Although the metal contents of plants and soil at a given site are not directly related, data from both types of samples indicate the presence of mineralization. Soil anomalies should be

displaced downslope from the site of mineralization through slumping and solifluction, and plant anomalies may be even further displaced through fluid flow. This may explain the extension of the plant gold anomaly from site 1 down to and including site 4, even though the soil gold anomaly is concentrated on sites 1, 2, and 3.

The Role of Inorganic Ligands

The low concentration of chloride in the surficial material suggests that it does not play a large role in the gold cycle in the Lillian Creek drainage. In addition, the pH of the soil is generally too high for the formation of significant amounts of gold chloride complexes. There is little correlation between chloride content of the soils and concentrations of water-extractable gold or gold in plants. Keeping in mind that water-extractable gold and plant gold are indicators of gold mobility, this relationship suggests that chloride is not closely linked to gold mobilization in the study area. These findings effectively eliminate chloride as an important part of the gold cycle in the study area.

Sulfur may be present in the soil in many forms, but generally will be oxidized to sulfate in the vadose zone. For this study, only the total sulfur content of the surficial material was determined, as the original sulfur compounds were most likely altered by sample preparation. The low total sulfur values suggest that thiosulfate is not likely to be an agent of gold mobilization in this area. In addition, the pH of the soil, which was generally below seven, indicates that thiosulfate would not be an intermediate product of sulfide oxidation (Goldhaber, 1983). A graph of total sulfur versus organic carbon shows a strong association with the organic fraction in the soil (fig. 6). This correlation implies that the sulfur in the soil is present mostly as organic compounds, as is expected in a noncalcareous soil (Biederbeck, 1978), and not as inorganic species such as thiosulfate. The chemical evidence does not support thiosulfate as a probable gold complexing agent under the conditions present in the Lillian Creek drainage.

Neither labrador tea nor alder is cyanogenic (cyanide-producing), and none of the other species of plants native to the study area are known to be cyanogenic (Severson and others, 1986). Cyanide was not detected in the soil samples by either of the methods used. Therefore, it is unlikely that cyanide plays a role in the mobilization of gold in the Lillian Creek drainage.

Distribution of Gold in the Soil

Water-extractable gold was more often detected in soil samples from the C-horizon than in those from the 0-horizon. This may be due to the leaching of relatively mobile forms of gold from the upper soil horizons into the lower soil horizons. Alternatively, it could be due to the presence of a greater abundance of gold-mobilizing agents in the C-horizon, or the adsorption of gold onto insoluble organic material in the 0-horizon.

The low quantity of water-extractable gold in the soils suggests that gold is not highly mobile under the current conditions present in the Lillian Creek drainage. However, the high levels of gold in some of the plants show that it is available. This discrepancy may be due to the uptake of gold by plants in some form that is not readily water-extractable. As the study by King and others (1984) and the experimental portion of this study demonstrates, even elemental

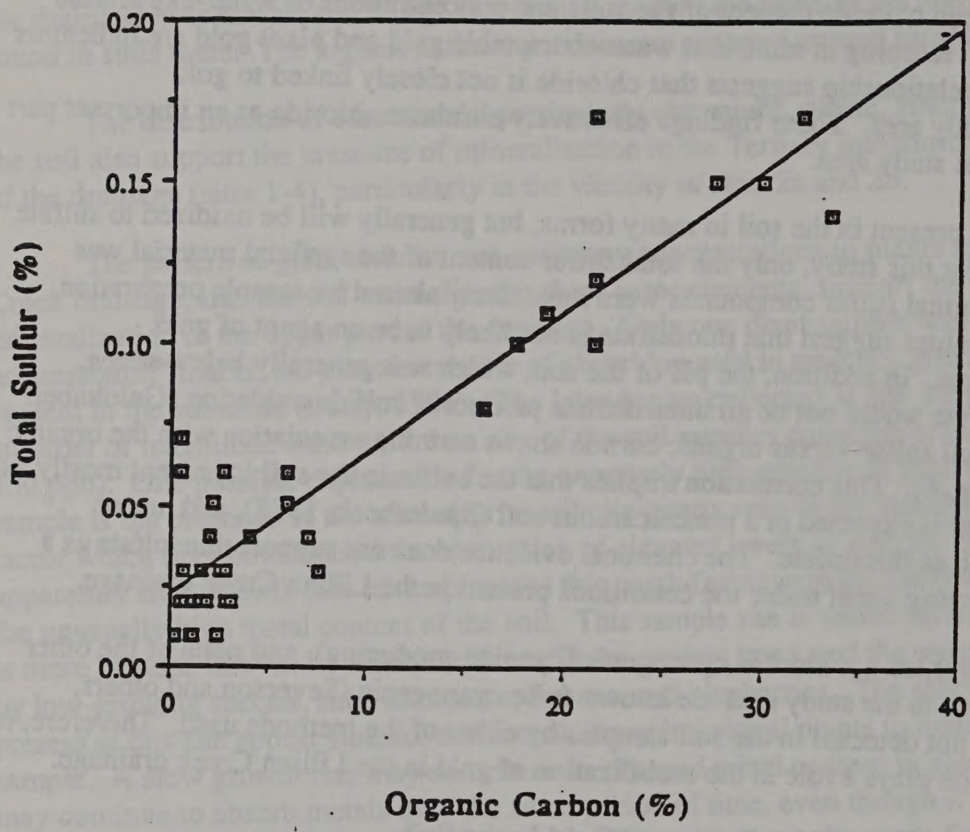


Figure 6.--Diagram showing the relationship between total sulfur and organic carbon in soil samples collected from Lillian Creek, Alaska.

gold can be made available to plants after a relatively brief exposure to moderately oxidizing conditions. Plants may absorb nutrients through a cation exchange process whereby they produce hydrogen to exchange with nutrients bound to particles in the soils (Malyuga, 1964; Brooks, 1983). These nutrients may not otherwise be available in the soil solution. Another reason for this discrepancy may be the incomplete recovery of water-extractable gold due to the readsorption of gold released during the extraction onto mineral and organic surfaces.

The gold which was measured in the water extracts may have been present in the soil in a variety of forms. These include colloids less than 0.45 µm, weakly adsorbed cations, or soluble complexes. The results of this study suggest that any soluble gold complexes are more likely to be organic than inorganic, due to a lack of inorganic ligands in the soil. The levels of loosely bound gold in the samples tested were approximately in the same range as water-extractable gold (0-10 ppb), and the two extractions may be identifying the same form or forms of gold. The water extraction was allowed to proceed overnight, however, while the ammonium acetate extraction was accomplished in two hours. Slight differences between the two data sets may have been caused by a lack of equilibration of the samples with the ammonium acetate.

A significant amount of gold is apparently associated with the organic fraction of the soil, present either as organic complexes or as cations adsorbed onto organic surfaces. The ratio of organic-associated gold to total gold decreases as total gold increases (fig. 7). There is little correlation between organic-bound gold and organic carbon in the soil. These observations suggest that the organic-bound gold is not in equilibrium with either elemental gold or organic material in the soil, and that the adsorption or complexation of gold by organics is controlled by nonequilibrium processes. Given the chemical stability of gold and the relative abundance of organic ligands in the soil, it is reasonable to assume that the oxidation of gold is the rate-controlling factor, and that organic ligands are always present in excess. It could be postulated that, given an infinite amount of time, equilibrium will be reached and the relationship between organic gold and total gold will be linear.

There is little correlation between organic-bound gold and water-extractable gold. This observation implies that, although a significant amount of organic-associated gold exists in the soil (up to 150 ppb), it is not highly mobile. It may be hypothesized that gold in this form is associated primarily with an insoluble organic fraction of the soil, such as the humin. If the organic-bound gold is sparingly soluble, it is possible that it may account for some of the gold measured in the water extracts.

The water-extractable organic carbon content of the soil parallels total organic carbon, and shows little correlation to water-extractable gold (table 6). These relationships neither support nor disprove the existence of soluble organic-gold species in the surficial material of the study area, and can be explained by either a lack of soluble organic-gold compounds in the soil or a disequilibrium relationship between dissolved organic carbon and gold, as was suggested for total carbon and gold.

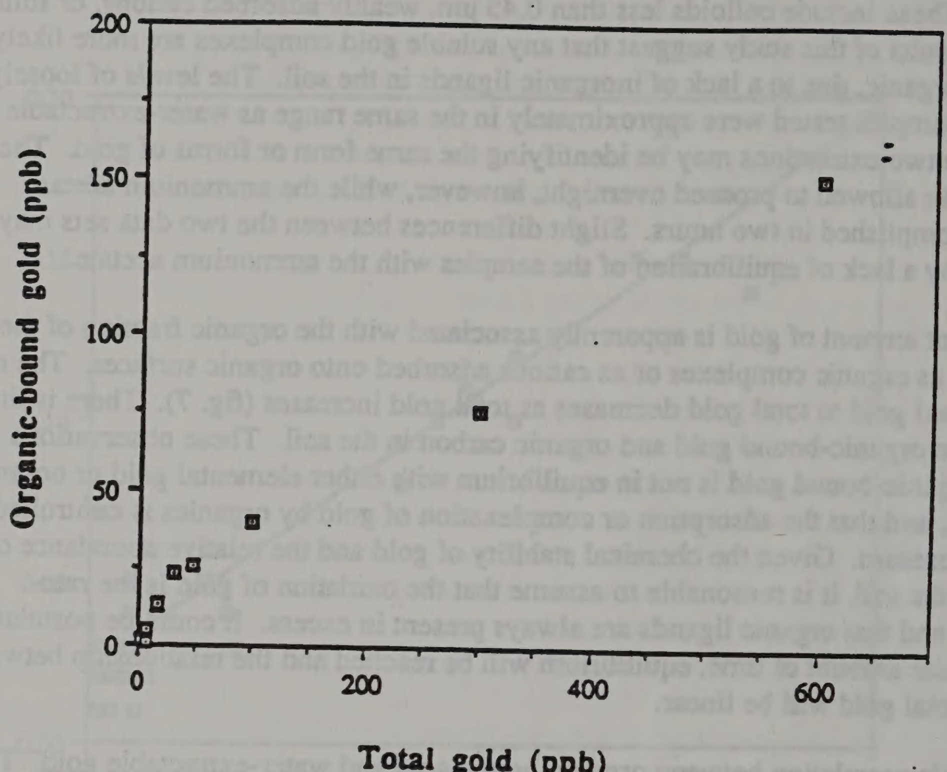


Figure 7.--Diagram showing the relationship between organic-bound gold and total gold in soils collected from Lillian Creek, Alaska.

Elemental Gold

The presence of crystalline gold in placer deposits has been used as evidence of the precipitation of gold from soil solutions at low temperatures (Warren, 1982; Severson and others, 1986). Many of the crystals from the Lillian Creek placer deposit observed in this study are very delicate and euhedral, and it is difficult to envision their deposition through physical displacement downstream without obvious damage. However, the crystals themselves cannot definitively indicate a chemical origin of some of the gold in the placer: the gold may escape deformation by weathering out of the rock very near to the place of its deposition, or through deposition by the slow process of solifluction (Severson and others, 1986).

Wilson (1984) documented the presence of euhedral "pseudo-trigonal" crystals in Western Australia which he stated were precipitated by groundwater in weathered mafic schists. These pseudo-trigonal crystals are crystallographically the same as the flattened octahedra which are found in the Lillian Creek placer deposits (fig. 3b). Trigonal and hexagonal crystals of supergene gold were noted in the weathering zone of the Fazenda Brasileiro deposit in Bahia, Brazil (Vasconcelos, 1987), as well as the Hannan South gold mine in Western Australia (Lawrence, 1988). Such crystals have also been formed experimentally by the reduction of dilute auric chloride solution at relatively low temperatures (99.5 °C) (Suito and Uyeda, 1953). Perhaps this morphology is typical of gold precipitated from low temperature aqueous solutions.

The striated and porous surface textures of some gold particles from the placer may be indicators of gold dissolution under conditions present in the surficial material of the placer. Severson and others (1986) found gold crystals from the Lillian Creek drainage with surface textures similar to the striated gold in figure 4c, and suggested that chemical etching may be the cause of this texture. The formation of the porous texture may also be the result of preferential leaching of silver from the nuggets as they weather, leaving a pitted surface (Mann, 1984).

The placer gold is of high fineness, containing less than 7.5 percent silver and quartz combined. Two of the samples contained greater than 97 percent gold. The nuggets from the placer exhibit the broadest range of chemical compositions, and may indicate that they are in various stages of weathering and, consequently, silver depletion (fig. 8) or from different sources. The compositions of the crystalline placer gold and the vein gold are similar and may suggest a common origin for both types of gold, but only two placer crystals were analyzed. The lower analytical totals of the vein gold may indicate the presence of an undetected element.

Factors Influencing Gold Uptake by Plants

Using the Spearman's rank correlation coefficient, several statistically significant correlations were found between plant and soil data. Due to the diverse nature of the surficial material in the placer, and the uncertainty of its origin, only correlations with the locally formed 0-horizon will be discussed.

Gold in labrador tea has a good positive correlation with the amount of organic carbon in the 0-horizon (fig. 9), as well as a negative correlation with pH. This relationship may be due to the formation of a gold complex, organic or inorganic, which is available to labrador tea under

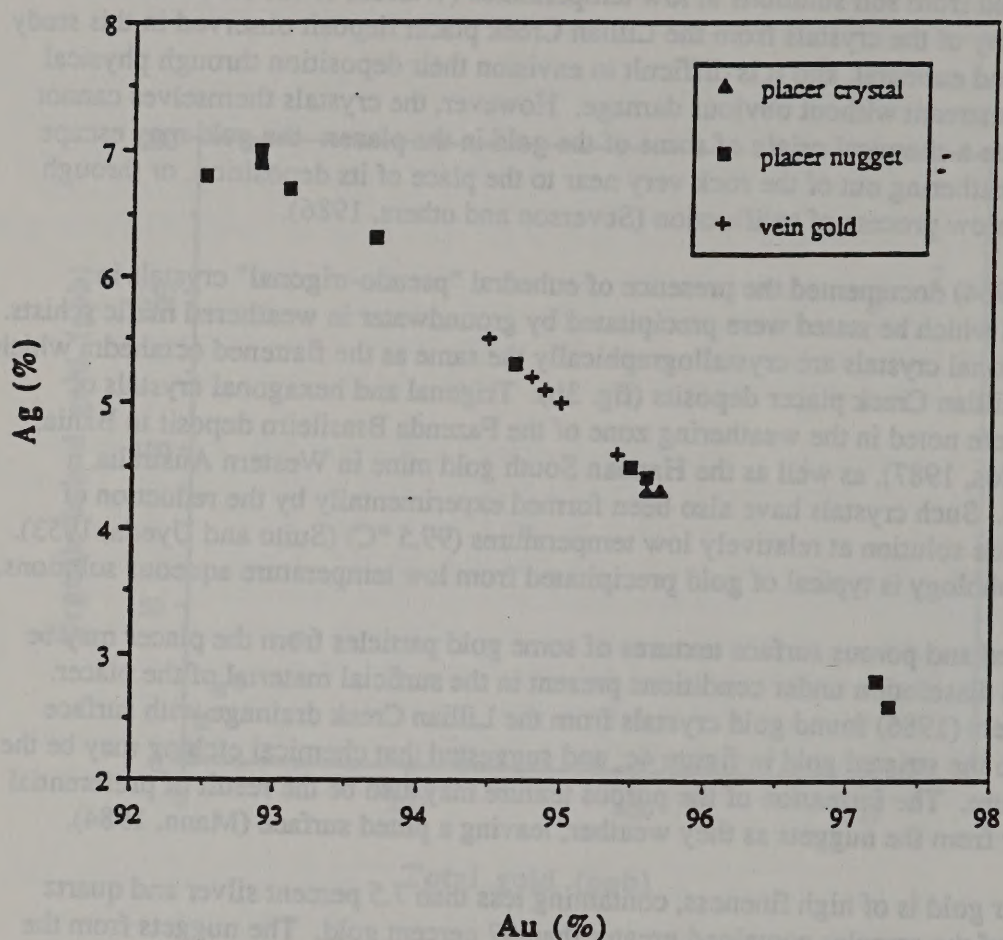


Figure 8.--Diagram showing the relationship between silver and gold for crystalline placer gold, placer nuggets, and vein gold from the Lillian Creek area, Alaska.

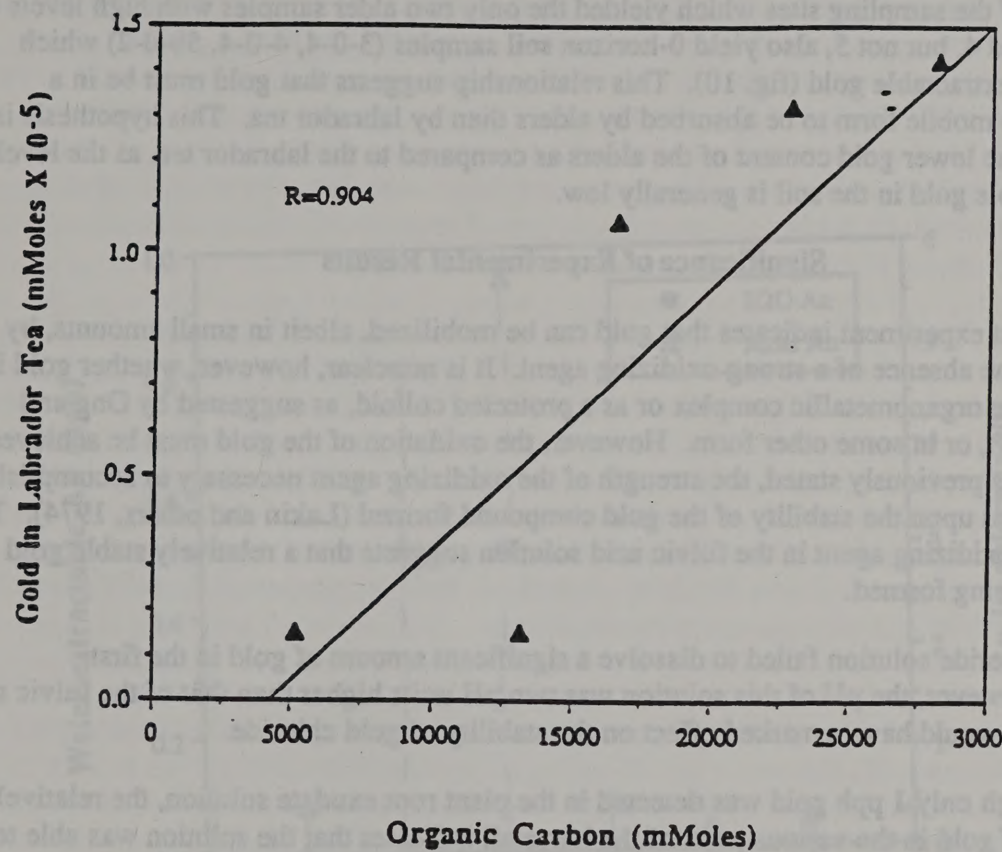


Figure 9.--Diagram showing the relationship between gold in labrador tea and organic carbon in the soil 0-horizon from Lillian Creek, Alaska.

the acidic conditions brought about by the presence of organic material. Because little correlation was found to exist between organic carbon and organic-bound gold in the soil, the relationship between plant gold and organic carbon probably is not due to the uptake of an organic-gold compound by labrador tea. There is no direct indication that the labrador tea is absorbing organic or inorganic gold species.

Most of the sampling sites which yielded the only two alder samples with high levels of gold, sites 3 and 4, but not 5, also yield 0-horizon soil samples (3-0-4, 4-0-4, 5b-0-2) which contain water-extractable gold (fig. 10). This relationship suggests that gold must be in a relatively more mobile form to be absorbed by alders than by labrador tea. This hypothesis is supported by the lower gold content of the alders as compared to the labrador tea, as the levels of water-extractable gold in the soil is generally low.

Significance of Experimental Results

The first experiment indicates that gold can be mobilized, albeit in small amounts, by fulvic acid in the absence of a strong oxidizing agent. It is not clear, however, whether gold is present as a true organometallic complex or as a protected colloid, as suggested by Ong and Swanson (1969), or in some other form. However, the oxidation of the gold must be achieved in any case, and as previously stated, the strength of the oxidizing agent necessary to accomplish this is dependent upon the stability of the gold compound formed (Lakin and others, 1974). The lack of strong oxidizing agent in the fulvic acid solution suggests that a relatively stable gold compound is being formed.

The chloride solution failed to dissolve a significant amount of gold in the first experiment; however, the pH of this solution was two pH units higher than that of the fulvic acid solution which would have a marked effect on the stability of gold chloride.

Although only 1 ppb gold was detected in the plant root exudate solution, the relatively high amount of gold in the various parts of the horsetail indicates that the solution was able to mobilize gold in some form. Cannon and others (1968) reported generally low values of gold in the ash of horsetail growing in mineralized areas, but they analyzed only the above-ground part of the plants. This study shows that the major concentration of gold in horsetail is located in the roots. Due to the difficulty of removing soil contamination from the roots, however, most biogeochemical surveys do not utilize plant roots.

The results of this experiment may be significant to the understanding of the uptake of gold by plants because it shows that gold may be dissolved and assimilated by plants in an oxidizing, organic system in a short period of time. This is in agreement with the findings of King and others (1984).

The results of the first experiment were believed to represent minimum values for gold dissolution in the natural environment because the single flakes of gold had limited surface area. In addition, the solubility of pure gold is generally lower than that of the gold-silver alloy which is usually found in nature. However, the gold content in the fulvic acid solution from the second experiment was even lower than in the first. Although the decrease in concentration of the fulvic acid solutions in the second experiment could account for some decrease in the amount of gold

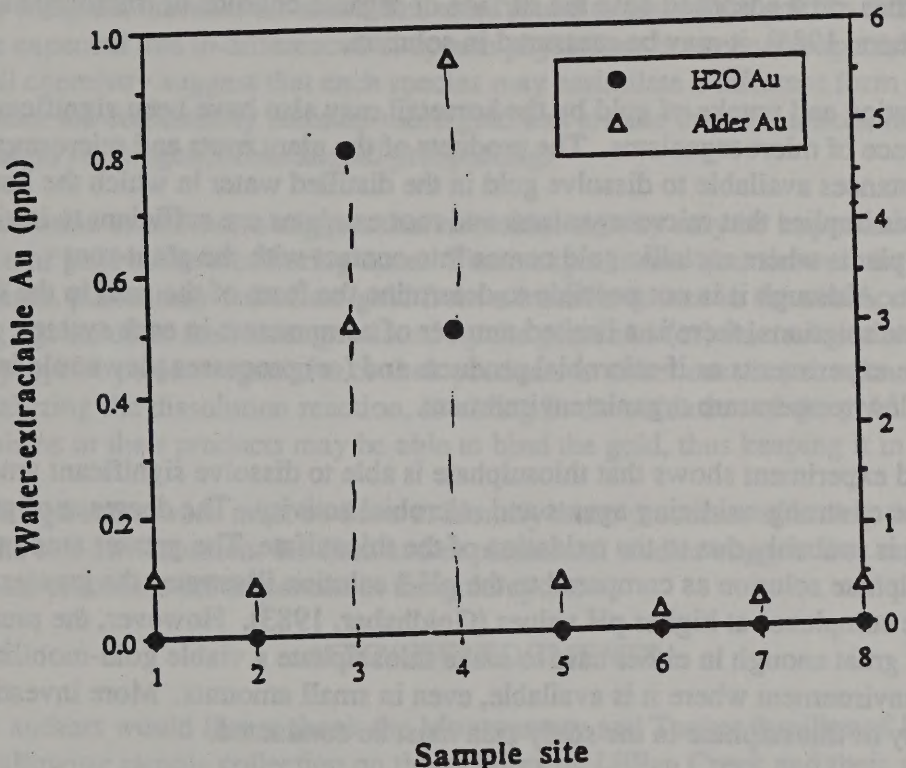


Figure 10.--Diagram showing the relationship between gold in alders and water-extractable gold in the soil 0-horizon at each sample site from Lillian Creek, Alaska.

mobilized, the increased surface area of the metallic gold would be expected to cause a much greater increase in gold dissolution.

One explanation for the greater amount of gold in the fulvic acid solution in the first experiment as compared to the second experiment is the presence of microbial activity in the solutions of the first experiment. It is possible that microbial activity in some way catalyzes the dissolution reaction, or oxidizes the gold, an essential step in the dissolution of gold. Both microbes and their products are capable of oxidizing metals in a variety of forms and by many mechanisms (Ehrlich, 1986), and it is reasonable that these processes may affect gold similarly. Once the gold is oxidized, however, a suitable "sink" must be available for the gold cations as gold is very unstable in ionic form. If the ionic gold forms complex with fulvic acid or microbial metabolites, or is adsorbed onto the surface of organic colloids or microorganisms (Watterson and others, 1985), it may be measured in solution.

The dissolution and uptake of gold by the horsetail may also have been significantly aided by the presence of microorganisms. The products of the plant roots and microorganisms were the only substances available to dissolve gold in the distilled water in which the horsetail were growing. This implies that microorganisms and root exudates are sufficient to initiate the uptake of gold by plants where metallic gold comes into contact with the plant-root microenvironment. Although it is not possible to determine the form of the gold in the fulvic acid or root exudate solutions, there is a limited number of components in each system. It appears from these experiments as if microbial products and (or) processes play a role in gold dissolution in the low temperature organic environment.

The second experiment shows that thiosulphate is able to dissolve significant amounts of gold in the absence of strong oxidizing agents and microbial activity. The decrease of gold in solution over time is probably due to the oxidation of the thiosulfate. The greater amount of gold in the pH 6 thiosulphate solution as compared to the pH 3 solution illustrates the greater stability of gold thiosulfate complexes at higher pH values (Goldhaber, 1983). However, the presence of gold in solution is great enough in either case to make thiosulphate a viable gold-mobilizing agent in any soil environment where it is available, even in small amounts. More investigations into the availability of thiosulphate in the study area must be conducted.

CONCLUSIONS

The geochemical and biogeochemical data identify an area of high gold concentration in the upper part of the Lillian Creek drainage. Other elements, particularly arsenic and some of the base metals, follow this distribution. These data support the hypothesis that the mineralization which is the source of the placer gold is localized at the head of the drainage.

The results of this study show that gold in the soil is not present entirely as the metal. An organic-bound form of gold accounts for more than half of the total gold in some soil samples. Although its exact nature is unknown, the organic-associated gold may be complexed with organic compounds or strongly adsorbed onto organic surfaces. This form of gold is not readily water-extractable, which implies that it is not highly mobile. It may be associated with humin, the insoluble organic fraction of the soil.

Small concentrations of water-extractable gold were found in the soil samples, suggesting that gold is being mobilized in the weathering zone. The form of this mobile gold is unknown, but this study has eliminated a few of the potential gold complexing agents. The data indicate that chloride and cyanide are not important in the cycling of gold in this environment. Although thiosulfate was shown by the experimental study to be an effective gold dissolving agent, it is not likely to be present in appreciable quantities in the slightly acidic soil of the placer deposit. Possible forms of the water-extractable gold include small colloids, weakly adsorbed ions, and soluble organic complexes.

Many of the plant samples taken from the Lillian Creek drainage were found to contain elevated levels of gold, even though very little water-extractable gold is present in the soils. This suggests that gold does not have to be highly mobile in order to be available to plants. The two species sampled, labrador tea and alder, were found to assimilate gold in different quantities, as might be expected due to differences in uptake physiology between the species. Correlations with the soil chemistry suggest that each species may assimilate a different form of gold. In this study, labrador tea consistently takes up more gold and arsenic than the alders, making it a more favorable target for biogeochemical gold prospecting.

Dissolution experiments suggest that microbial activity may be important in the mobilization of gold in the weathering zone. These experiments also showed that, given a short amount of time, plants may assimilate gold from a metallic source in the absence of any complexing agents other than those produced by the plant and (or) microorganisms. The precise role that is played by microorganisms or their products in either case is yet unknown, but may involve catalyzing the dissolution reaction, or aiding in the oxidation of the gold. The microorganisms or their products may be able to bind the gold, thus keeping it in solution.

Although more work must be done to identify exact processes affecting the geochemistry of gold in the soil environment, the field and experimental studies suggest that organic material and biological processes are important in the gold cycle.

ACKNOWLEDGMENTS

The authors would like to thank the Montgomery and Tucker families of Livengood, Alaska for allowing sample collection on their claims on Lillian Creek and their suggestions during field work. The majority of the analytical work was performed by Katie Walton-Day, Elaine Brandt, Paul Briggs, Thomas Peacock, Randall Hill, John McHugh, and Eric Welsch, all of the Branch of Geochemistry, Denver, Colorado. Thanks are due to Phil Bennett, Robert Falk, and Lynton Land, University of Texas at Austin, for their advice, encouragement, and reviews. Partial support for this study is acknowledged from the University of Texas Geology Foundation.

REFERENCES CITED

Aiken, G.R., 1985, Isolation and concentration techniques for aquatic humic substances, *in* Aiken, G.R. and others, eds., Humic substances in soil sediment, and water: New York, John Wiley and Sons, 692 p.

Albanese M.D., 1983a, Geochemical reconnaissance of the Livengood B-3, B-4, C-3, and C-4 quadrangles, Alaska--Summary of data on stream sediment, pan-concentrate, and rock samples: Alaska Division of Geological and Geophysical Surveys Report of Investigations 83-1, 55 p.

Andrade, W.O., Machesky, M.L., and Rose, A.W., 1988, Interactions of gold with humic substances [abs.]: Geological Society of America Abstracts with Programs, p. A240.

Baker, W.E., 1978, The role of humic acid in the transport of gold: Geochim. et Cosmochim., v. 42, p. 645-649.

Berry, W.L., 1986, Plant factors influencing the use of plant analysis as a tool for biogeochemical prospecting, *in* Carlisle, D. and others, eds., Mineral exploration: Biological systems and organic matter, Rubey Volume V: Englewood Cliffs, Prentice-Hall, p. 13-32.

Biederbeck, V.O., 1978, Soil organic sulfur and fertility, *in*Schnitzer, M., and Khan, S.U., eds., Developments in soil science 8: Soil organic matter: Amsterdam, Elsevier Scientific Publishing Company, p. 273-310.

Boyle, R.W., 1979, The geochemistry of gold and its deposits: Geological Survey of Canada Bulletin 280, 584 p.

Brooks, A.H., 1916, Preliminary report on the Tolovana district, Alaska: U.S. Geological Survey Bulletin 642, p. 201-209.

Brooks, R.R., 1983, Biological methods of prospecting for minerals: New York, John Wiley & Sons, 322 p.

Cannon, H.L., Shacklette, H.T., and Bastron, H., 1968, Metal absorption by Equisetum (horsetail): U.S. Geological Survey Bulletin 1278-A, 21 p.

Chao, T.T., 1984, Use of partial dissolution techniques in geochemical exploration: Journal of Geochemical Exploration 20, p. 101-135.

Cohen, D.R., Hoffman, E.L., and Nichol, I., 1987, Biogeochemistry: A geochemical method for gold exploration in the Canadian shield: Journal Geochemical Exploration, v. 29, p. 49-73.

Crock, J.G., and Lichte, F.E., 1982, An improved method for the determination of trace-level arsenic and antimony in geological materials by automated hydride generation-atomic absorption spectroscopy: Analytica Chemica Acta, v. 144, p. 223-233.

Crock, J.G., Lichte, F.E., and Briggs, P.H., 1983, Determination of elements in National Bureau of Standards Geological reference materials SRM 278 Obsidian and SRM 688 Basalt by inductively coupled argon plasma-atomic emission spectrometry: Geostandards Newsletter, v. 7, p. 335-340.

Crock, J.G., and Severson, R.C., 1980, Four reference soil and rock samples for measuring element availability in the western energy regions: U.S. Geological Survey Circular 841, 16 p.

Curtin, G.C., Lakin, H.W., and Hubert, A.E., 1970, The mobility of gold in mull (forest humus layer): U.S. Geological Survey Professional Paper 700-C, p. 127-129.

Dexter-Dyer Grosovsky, B., 1983, Microbial role in Witwatersrand gold deposition, *in* Westbroek, P., and deJong, E.W., eds., Biomineralization and biological metal accumulation, p. 495-498.

Eakins, G.R., 1974, Preliminary investigations, Livengood mining district, Alaska: Alaska Div. Geol. Geophys. Surveys Open-File Report 40, 16 p.

Ehrlich, H.L., 1986, Interactions of heavy metals and microorganisms, *in* Carlisle, D. and others, eds., Mineral exploration: Biological systems and organic matter, Rubey Volume V: Englewood Cliffs, Prentice-Hall, p. 221-237.

Fedoseyeva, V.I., Fedoseyev, N.F., and Zvonareva, G.V., 1985, Interaction of some gold complexes with humic and fulvic acids: Geokhimiya, no. 9, p. 1386-1390.

Fetzer, W.G., 1934, Transportation of gold by organic solutions: Economic Geology, v. 29, p. 599-604.

Foster, R.L., 1966, The petrology and structure of the Amy Dome area, Tolovana mining district, east-central Alaska: unpublished Ph.D. thesis, University of Missouri, 227 p.

Foster, R.L., 1968, Descriptions of the Ruth Creek, Lillian Creek, Griffin, Old Smoky, Sunshine No. 2, and Olive Creek lode prospects, Livengood district, Alaska: U.S. Geological Survey Open-File Report ____, 21 p.

Foster, R.L., 1969, Potential for lode deposits in the Livengood gold placer district, east-central Alaska: U.S. Geological Survey Circular 590, 18 p.

Freise, F.W., 1931, The transport of gold by organic underground solutions: Economic Geology, v. 26, no. 4, p. 421-431.

Goldhaber, M.B., 1983, Experimental study of metastable sulfur oxyanion formation during pyrite oxidation at pH 6-9 and 30 °C: American Journal of Science, v. 283, p. 193-217.

Gosling, A.W., Jenne, E.A., and Chao, T.T., 1971, Gold content of natural waters in Colorado: Economic Geology, v. 66, p. 309-313.

Gregoire, D.C., 1985, Selective extraction of organically bound gold in soils, lake and stream sediments: J. Geochem. Explor., v. 23, p. 299-313.

Hare, F.K., and Hay, J.E., 1974, The climate of Canada and Alaska, *in* Landsberg, H.E., ed., World Survey of Climatology, v. 11: Amsterdam, Elsevier Scientific Publishing Company, p. 49-192.

Helgeson, H.C., and Garrels, R.M., 1968, Hydrothermal transport and deposition of gold: Economic Geology, v. 63, no. 2, p. 622-635.

Henley, R.W., 1973, Solubility of gold in hydrothermal chloride solutions: Chemical Geology, v. 11, p. 73-87.

Huheey, J.E., 1983, Inorganic chemistry: Principles of structure and reactivity [3d ed.]: New York, Harper & Row, 936 p.

Jackson, L.L., Brown, F.W., and Neil, S.T., 1987, Major and minor elements requiring individual determination, classical whole rock analysis, and rapid rock analysis, *in* Baedecker, P.A., ed., Methods for geochemical analysis: U.S. Geological Survey Bulletin 1770, p. G1-G23.

Johnson, B.F.G., and Davis, R., 1973, Gold, *in* Bailor, J.C. and others, eds., Comprehensive inorganic chemistry: Oxford, Pergamon Press, Inc., p. 129-186.

Jones, K.C., and Peterson, P.J., 1989, Gold uptake by perennial ryegrass--the influence of humates on the cycling of gold in soils: Biogeochemistry, v. 7, p. 3-10.

Krauskopf, K.B., 1951, The solubility of gold: Economic Geology, v. 46, p. 858-870.

King, H.D., Curtin, G.C., and Shacklette, H.T., 1984, Metal uptake by young conifer trees: U.S. Geological Survey Bulletin 1617, 23 p.
Lakin, H.N., Curtin, G.C., and Hubert, A.E., 1974, Geochemistry of gold in the weathering cycle: U.S. Geological Bulletin 1330, 50 p.
Lawrence, L.M., 1988, The morphology and geochemistry of supergene gold at Hannan South gold mine, Western Australia, *in* Bicentennial gold '88 extended abstracts and posters programme: Geological Society of Australia, Inc., Abstracts no. 23, v. 1, p. 360-364.
Levitskiy, V.V., Razvozzhayeva, E.A., Demin, B.G., Khrenov, P.M., Baburin, L.M., and Odintsova, I.V., 1983, Asfal'teny neftey i zolotykh rudkontsentratory metalloorganicheskikh soyedineniy: Doklady Akademii Nauk SSSR, v. 270, no. 1, p. 196-199.
Malyuga, D.P., 1964, Biogeochemical methods of prospecting: Consultants Bureau, New York, 205 p.
Mann, A., 1984, Mobility of gold and silver in lateritic weathering profiles--Some observations from Western Australia: Economic Geology, v. 79, p. 38-49.
McHugh, J.B., 1984, Gold in natural waters--A method of determination by solvent extraction and electrothermal atomization: J. Geochem. Explor., v. 20, p. 303-310.
Mertie, J.B., Jr., 1918, The gold placer of the Tolovana district, Alaska: U.S. Geological Survey Bulletin 662-D, p. 221-277.
Mertie, J.B., Jr., 1937, The Yukon-Tanana region, Alaska: U.S. Geological Survey Bulletin 872, 276 p.
Mineyev, G.G., 1976, Organisms in the gold migration-accumulation cycle: Geokhimiya, no. 4, p. 577-582.
Munsell Soil Color Charts, 1975 ed., Macbeth Division of Kollmorgen Corporation, Baltimore, Maryland.
Nelson, D.W., and Sommers, L.E., 1982, Total carbon, organic carbon, and organic matter, *in* Page, A.L., ed., Methods of soil analysis, Part 2: Chemical and microbiological properties, [2d ed.]: Madison, Wisconsin, ASA-SSSA, p. 539-579.
Nemec, B., Babicka, J., and Oborsky, A., 1936, Uber das vorkommen von gold in den schachtelhalmen [On the accumulation of gold in horsetails]: International Academy of Science Boheme Bulletin, nos. 1-7, 13 p.
Ong, H.L., and Swanson, V.E., 1969, Natural organic acids in the transportation, deposition, and concentration of gold: Quarterly of the Colorado School of Mines, v. 64, p. 395-425.
Overbeck, R.M., 1920, Placer mining in the Tolovana district, Alaska: U.S. Geological Survey Bulletin 712, p. 177-184.
Pearson, R.G., 1973, Hard and soft acids and bases: Stroudsburg, Pennsylvania, Dowden, Hutchinson & Ross, 302 p.
Péwé, T.L., 1952a, Geomorphology of the Fairbanks area: Ph.D. thesis, Stanford University, 220 p.
Péwé, T.L., 1975, Quaternary geology of Alaska: U.S. Geological Survey Professional Paper 835, 145 p.
Rashid, M.A., and Leonard, J.D., 1973, Modifications in the solubility and precipitation behavior of various metals as a result of their interaction with sedimentary humic acid: Chemical Geology, v. 11, p. 89-87.
Renders, P.J., and Seward, T.M., 1989, The stability of hydrosulphido- and sulphido- complexes of Au(I) and Ag (I) at 25 °C: Geochim. et Cosmochim. Acta, v. 53, p. 245-253.

Robinson, M.S., 1983, Bedrock geologic map of the Livengood C-4 quadrangle, Alaska: Alaska Division of Geological and Geophysical Surveys Report of Investigations 83-4, 1 sheet, scale 1:40,000.

Saunders, R.H., 1958, Notes on the mineral resources of Livengood Creek, Hess Creek, and their tributaries: Territory of Alaska Department of Mines, MR 49-3, 3 p.

Schnitzer, M., 1978, Humic substances--Chemistry and reactions, *in* Schnitzer, M., and Khan, S.U., eds., Developments in soil science 8: Soil organic matter: Amsterdam, Elsevier Scientific Publishing Co., p. 1-64.

Severson, R.C., Crock, J.G., and McConnell, B.M., 1986, Processes in the formation of crystalline gold in placers, *in* Dean, W.E., ed., Proceedings of the Denver region Exploration Geologists Society symposium: Organics and ore deposits, p. 69-80.

Seward, T.M., 1973, Thio complexes of gold and the transport of gold in hydrothermal solutions: Geochim. Cosmochim. Acta, v. 37, p. 379-399.

Seward, T.M., 1982, The transport and deposition of gold in hydrothermal systems, *in* Foster, R.P., ed., Gold '82--The geology, geochemistry, and genesis of gold deposits: Rotterdam, A.A. Balkema, p. 165-181.

Shacklette, H.T., Lakin, H.W., Hubert, A.E., and Curtin, G.L., 1970, Absorption of gold by plants: U.S. Geological Survey Bulletin 1314-B, 23 p.

Sigafoos, R.S., 1958, Vegetation of northwestern North America as an aid in the interpretation of geologic data: U.S. Geological Survey Bulletin 1061-E, p. 165-185.

Smith, P.S., 1933, Past placer gold production from Alaska: U.S. Geological Survey Bulletin 857-B, p. 92-98.

Smith, T.E., 1983, Bedrock geologic map of the Livengood C-3 quadrangle, Alaska: Alaska Division of Geological and Geophysical Surveys Report of Investigations 83-5, 1 sheet, scale 1:40,000.

Stevenson, F.J., 1982, Humus chemistry--Genesis, composition, reactions: New York, John Wiley & Sons, 443 p.

Suito, E., and Uyeda, N., 1953, Study of single micro-crystals of gold by a three-stage electron microscope: Proc. Japan Academy, v. 29, no. 7, p. 324-330.

Thompson, C.E., Nakagawa, H.W., and VanSickle, G.H., 1968, Rapid analysis for gold in geological materials: U.S. Geological Survey Professional Paper 600-B, p. 130-132.

Thornton, I., 1986, Soil and plant factors that influence element availability and uptake--Implications for geochemical prospecting, *in* Carlisle, D. and others, eds., Mineral exploration--Biological systems and organic matter, Rubey Volume V: Englewood Cliffs, Prentice-Hall, p. 47-59.

United States Environmental Protection Agency, 1986, Test methods for evaluating solid waste--Physical and chemical methods, v. 1C, [3d ed.]: NTIS SW846, Springfield, Virginia, 218 p.

Van Dijk, H., 1971, Cation binding of humic acids: Geoderma, v. 5, p. 53-67.

Vasconcelos, P.M., 1987, Gold geochemistry in a semiarid weathering environment--A case study of the Fazenda Brasileiro deposit, Bahia, Brazil: unpublished M.A. thesis, Austin, University of Texas, 254 p.

Vinogradov, A.P., 1959, The geochemistry of rare and dispersed chemical elements in soils [2d ed., revised]: New York, Consultants Bureau, Inc., 209 p.

Warren, H.V., 1982, The significance of a discovery of gold crystals on overburden, *in* Levinson, A.A., ed., Precious metals in the Northern Cordillera: The Association of Exploration Geochemists, p. 45-51.

Watterson, J.R., Nishi, J.M., and Botinelly, T., 1985, Evidence that gold can nucleate on bacterial spores, *in* Bartsch-Winkler, S., and Reed, K.M., eds., The United States Geological Survey in Alaska: Accomplishments during 1983: U.S. Geological Survey Open-File Report 82-1004, p. 25-26.

Watterson, J.R., 1985, Crystalline gold in the soil and the problem of supergene nugget formation--Freezing and exclusion as genetic mechanisms: Precambrian Res., v. 30, p. 321-335.

Weber, F.R., Smith, T.E., Hall, M.H., and Forbes, R.B., 1985, Geologic guide to the Fairbanks-Livengood area, east-central Alaska: Anchorage, Alaska Geological Society, 44 p.

Webster, J.G., 1986, The solubility of gold and silver in the system Au-Ag-S-02-$H_2$0 at 25 °C and 1 atm: Geochim. et Cosmochim. Acta, v. 50, p. 1837-1845.

Wilson, A.F., 1984, Origin of quartz free nuggets and supergene gold found in laterites and soils--A review and some new observations: Australian Journal of Earth Sciences, v. 31, p. 303-316.

Economic Geolog
Vol. 66, 1971, p

Geological Features of Alluvial Placers

I. P. Kartashov

Abstract

Alluvial placer deposits may be divided into autochthonous and allochthonous subtypes. Autochthonous placers contain large heavy mineral grains that are practically immovable by streams and thus occur adjacent to primary ore deposits. Rich "bottom" autochthonous placers accumulate during many stages of river development and are concentrated at the base of the alluvium or in the crevices of its bedrock. Smaller and poorer "above-bottom" autochthonous placers are composed of grains received by rivers during the last stages of equilibrium and/or aggradation. These grains cannot get to the bedrock, which is protected from the river action by a layer of a substrative alluvium, but mineral grains may be concentrated in the higher parts of alluvial strata. During later downcutting stages, however, above-bottom placers may be reworked to join and enlarge bottom placers. Allochthonous placers, containing finer heavy mineral grains moved by a river as a part of its alluvial load, form far downstream from primary ore deposits or autochthonous placers, and they occur in surficial parts of channel alluvium. Rivers in an equilibrium state build point-bar placers, and in aggrading rivers several point-bar concentrations may be superimposed to form thick allochthonous placers.

As bottom and above-bottom autochthonous placers, and allochthonous placers, require different methods for locating, prospecting, and sampling, the recognition of these types is of a practical importance.

Introduction

In the Soviet Union, the geology of placer deposits has been studied not only in its practical aspects, but also from theoretical points of view. Placers of different origin and mechanisms of formation have been thoroughly characterized by Y. A. Bilibin in his "Principles of placer geology" (1938), which has had several editions and has become a textbook for placer geologists. However, some new concepts of placer geology, especially those concerning alluvial placers, have been developed since 1938. This paper, which systematically characterizes the main geological features of alluvial placers, is based on a revision of Bilibin's views and on later ideas of Soviet geologists, including those of the author.

Autochthonous and Allochthonous Alluvial Placers

Y. A. Bilibin pointed out that alluvial placers can be classed according to their connections with certain land forms, but a more basic distinction into two subtypes may be made on the basis of the different behavior of the "placer" minerals in flowing water. Where the heavy minerals are carried as a part of the bed load of the stream "bed placers" are formed; where they are carried in a suspended or semisuspended state "point-bar placers" are deposited. Bed placers, including channel, valley, and terrace placers, consist of relatively coarse mineral grains concentrated at the base of alluvial horizons and in the crevices of an underlying bedrock. Point-bar placers consist of considerably finer mineral grains concentrated in surficial parts of alluvial horizons and occur downstream of "bed" placers. Delta placers have been considered by Bilibin as "point-bar placers which have reached areas of accumulation."

This division of alluvial placers into these two subtypes reflects important features of their formation. It also helps one to understand the origin of these features and therefore to direct the search for placers more effectively. However, recent data on the structure of placers make it impossible to accept Bilibin's explanation of the differences between the subtypes without modification.

We now recognize that point-bar placers are never concentrated in floodplain (overbank) and ox-bow facies, which consist mainly of material carried by streams in suspended state. Even the point-bar placers of such a comparatively light mineral as diamond occur in the channel alluvium only. As the channel alluvium consists of bed-load material (Shantser, 1951), the sharp differences between point-bar and "bed" placers evidently cannot be explained by differences in means of transportation of mineral grains. They seem to be determined by the fact that "bed" placers contain concentrations of mineral grains which *practically are not displaced by flowing water*.

Extremely small mobility for placer gold was suggested by P. K. Yavorovsky as early as 1896. In 1949, N. A. Shilo expressed, and later (Shilo, 1956) argued in detail, an opinion that gold of "bed"

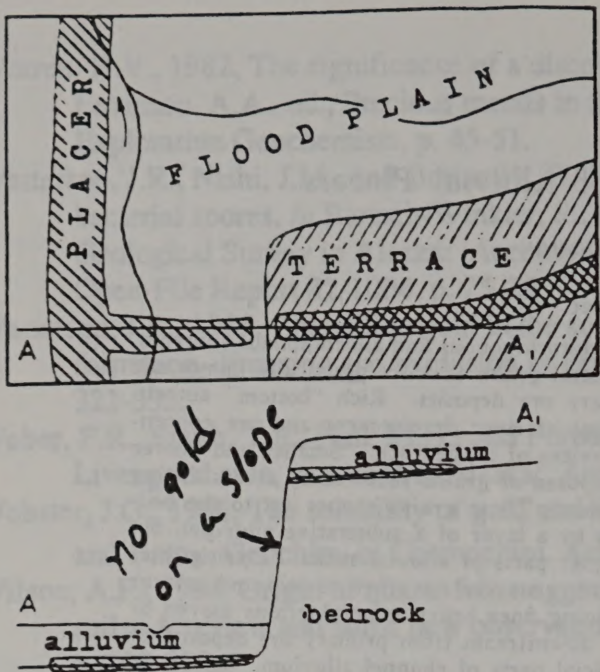


Fig. 1. A tributary placer continued in the main river.

placers, free from the country rock, is not displaced by water streams. The same opinion has been voiced independently by N. G. Bondarenko (1957). It is based mainly on the fact that some gold placers stretch across river valleys, being, as it were, continuations of terrace placers situated in the tributaries of these valleys (Fig. 1). Such cases can only result from a destruction of the lower courses of these tributaries and a redeposition of their placers by the main river. Configurations of such redeposited placers show that gold has not been displaced downstream by the main river, though the height of terraces, and therefore a vertical displacement of gold, has sometimes been 25 meters or more.

Placers of the preserved parts of tributaries occur not only on terraces but also on the bedrock underlying valley floors of the tributaries. If downcutting of the tributaries is accompanied by lateral migration, "breaks" may be formed in placers, as shown in Figure 2. Such "breaks," especially those similar to the "break" shown at C, can be satisfactorily explained only if the minerals of "bed" placers are virtually undisplaced by the main river, although displaced rather considerably by slope denudation.

Bilibin's terms "bed" and "point-bar," therefore, seem unsuitable for the two main subtypes of alluvial placers, as they do not properly reflect the mechanism responsible for their formation. I suggest calling them "autochthonous" and "allochthonous," as these generally accepted geological terms, when applied to alluvial placers, clearly define the main differences in their origin and do not require explanation.

Formation of Autochthonous Placers

According to N. A. Shilo (1956) autochthonous gold placers extending along river valleys are a result of displacement and gradual disintegration of pebbles with gold inclusions. In his opinion, gold released from such pebbles during their travel along the rivers accounts for the main part of placer gold. In my opinion, some gold particles undoubtedly are supplied to placers by the disintegration of ore pebbles, but such particles do not account for most placer gold. The changes of gold content in placers are influenced by peculiarities in the shape of the bedrock surface, and changes of mean dimensions of gold particles along the valleys are regular rather than casual, with particle size decreasing downstream. These facts cannot be explained without assuming that displacement of some gold by streams does take place inside the autochthonous placers. Thus, there is no need to explain the extension of autochthonous placers along the valleys by transportation of gold in pebbles. In fact, according to an old concept of Bilibin, destruction of alluvial pebbles occurs mainly by gradual attrition, and gold particles thus released are so fine that streams easily move them along, concentrating them only in allochthonous placers.

Placer gold may be divided not only into two groups of fractions, forming autochthonous and allochthonous placer occurrences, but according to A. V. Khripkov (1958), the gold of autochthonous placers may be further divided into a coarser "passive" fraction, which is practically immovable by streams, and a finer "active" fraction, which moves slowly down the river. The three fractions are transitional, and their division is only a scheme to help understand the mechanism of placer formation. Boundaries between them are dependent upon hydrodynamic characteristics. For example, gold particles finer than 1 mm form part of "passive" fractions in some placers, whereas gold particles up to 3 mm (Khripkov, 1958) may be a part of the "active" fractions of others placers.

Thus, the downstream extension of placers results from displacement of "active" fractions of mineral grains by water and from the fact that even "passive" fractions are not absolutely immovable. During river downcutting of even as much as tens of meters, the extremely small mobility of "passive" fractions makes their lateral displacement practically imperceptible and allows them to accumulate in valleys during many such downcuttings, extending perhaps even as long as entire geological periods. Each downcutting is accompanied by a total rewashing of alluvium and settling of placer minerals to the bedrock, where they

...in the placer minerals accumulated earlier. If all heavy minerals could be readily moved by streams, long-term accumulation of these minerals and formation of truly rich placers would be impossible. On the other hand, as rich placers form during many stages of river development that may include many downcuttings amounting to kilometers, and possibly even tens of kilometers, those insignificant downstream migrations of "passive" fractions that are imperceptible during a single downcutting may ultimately amount to a considerable distance. Thus, displacement of "passive" fractions has also to be reckoned among the causes of the extension of autochthonous placers along the river valleys.

"Bottom" and "Above-Bottom" Autochthonous Placers

V. V. Lamakin (1948) divided alluvial deposits into instrative, perstrative, and constrative dynamic phases, corresponding to the stages of river downcutting, dynamic equilibrium, and aggradation.[1] respec- in the mode of occurrence, lithologic characteristics, and distribution of different alluvial placers, but although Lamakin's concept must be considered by geologists studying alluvial placers his ideas need some modification.

The transition from downcutting to the stage of dynamic equilibrium is accompanied by formation of a so-called "alluvium of normal thickness." Only the upper strata of this alluvium are rewashed by a river in an equilibrium state, as the lower strata occur below the level of maximum depths of the river and are, therefore, protected. Thus, an alluvium of normal thickness consists of two strata formed under different conditions. These strata differ in age, lithologic characteristics, and other features, the differences being of the same order as those between dynamic phases (Kartashov, 1965). Evidently, only the upper strata may be regarded as the perstrative alluvium, and the lower strata should be referred to as a subphase of alluvium formed by an independent dynamic phase. I have suggested it be called the substrative alluvium.[2]

The long-term accumulation of "placer" minerals forms autochthonous placers concentrated at the base of instrative (downcutting stage) or substrative

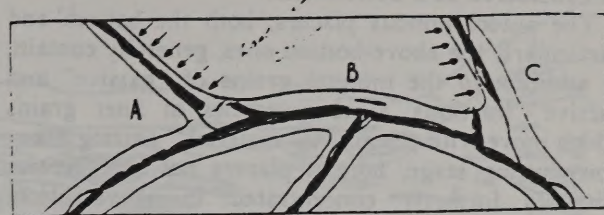


FIG. 2. "Breaks" in placers resulting from their redeposition under combined influence of river downcuttings and slope denudation. Arrows show the direction of lateral displacements of tributaries.

A—The lateral displacement of a tributary directed up the main river.

B—No pronounced lateral displacement of a tributary and no "break" in a placer.

C—The lateral displacement of a tributary directed down the main river.

(equilibrium and aggradation stage) alluvium and in the crevices of bedrock. As rivers come into contact with the bedrock bottom only during downcutting, bedrock placers receive new portions of minerals only during this stage. Minerals getting into rivers from exposed ore deposits or terrace placers during an equilibrium and/or aggradation stage cannot get into substrative alluvium and join the placers concentrated therein. However, they can form autochthonous concentrations at the base of the perstrative alluvium and/or within the constrative series. Such concentrations form during relatively short periods and are, as a rule, smaller and poorer than the placers of a long-term accumulation. Thus, autochthonous placers consist of two groups which I have called "bottom" and "above-bottom" placers (Fig. 3).

It should be emphasized that the position of a placer relative to bedrock is not a decisive characteristic of either group. The distinguishing feature of bottom placers is the presence of mineral grains received during many stages of river development, whereas above-bottom placers contain mineral grains received during the last equilibrium stage or during periods of the last aggradation stage. For example, the downcutting of a river through the thick alluvial strata formed during several alternating stages of aggradation and equilibrium may cease when the bed-

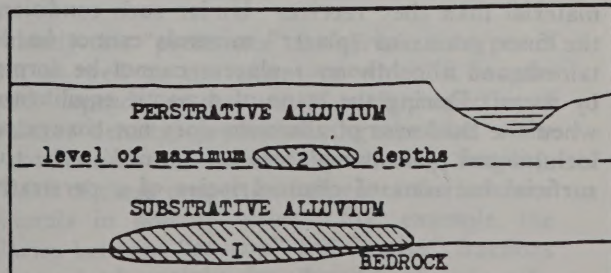


FIG. 3. Position of (1) bottom and (2) above-bottom placers in an alluvium of a normal thickness.

[1] The downcutting and aggradation stages correspond to the long-term fluvial processes defined by J. H. Mackin (1948). The Russian equivalent of a term "aggradation" is seldom used. As a rule, we call that stage of river development an "accumulation of alluvium," using both a Russian "nakoplenie" and russified Latin "akkumulatsia."

...y. This division reflects substantial differences

[2] The existence of a substrative alluvium, which does not permit a river in an equilibrium state to come into contact with the bedrock bottom, shows that such a phenomenon as "shifting" equilibrium accompanied by a degradation of rivers (Mackin, 1948) does not, as a matter of fact, exist.

rock bottom is not yet reached. If the rewashed strata had contained above-bottom placers, these placers could be redeposited together to form a rich placer on the "false bedrock"; such a placer ought to be considered as a bottom one.

The autochthonous placers, both the bottom and particularly the above-bottom ones, generally contain, in addition to the mineral grains of "passive" and "active" fractions, some quantities of finer grains which move with the alluvial material. During every downcutting stage, bottom placers not only receive minerals formerly concentrated in above-bottom placers or scattered in alluvium and in colluvial deposits of valley sides, but also they lose finer grains and some quantities of "active" fractions that are carried out of the concentration zones of the autochthonous placers. While the placers are receiving more minerals than they lose, they are increasing. and with the opposite ratio they are decreasing. However, the bottom placers may continue to exist, though gradually decreased, at the expense of "passive" fractions, during many stages of river development even after the complete destruction of their ore sources.

Formation of Allochthonous Placers

The mineral grains carried out of the concentration zones of autochthonous placers cannot accumulate to form deposits of the same kind, and they form dispersion zones. However, they may form another kind of mineral concentration, termed allochthonous placers. In contrast to the autochthonous placers, which form under practically any hydrodynamic conditions providing the rivers receive enough large mineral grains from the ore sources, the allochthonous placers require definite hydrodynamic conditions for their formation. Allochthonous placers are formed on the surfaces of point-bars and channel bottoms where the rivers carrying loose material with finer grains of "placer" minerals regularly deposit their load but also regularly remove the lighter minerals and rock fragments. This process leads to formation of surficial concentrations of heavy "placer" minerals. Such concentrations cannot originate if the water does not remove sufficient quantities of light material, or if, on the contrary, it removes both light material and considerable quantities of "placer" minerals.

Downcutting parts of rivers carry out more loose material than they receive. Under such conditions, the finer grains of "placer" minerals cannot be detained, and allochthonous placers cannot be formed by them. During the stage of dynamic equilibrium, when the thickness of alluvium does not change, allochthonous point-bar placers are formed in the thin surficial horizons of channel facies of a perstrative alluvium. Most favorable seems to be the stage of aggradation, when new point-bar concentrations may be superimposed on older ones, thus forming thick allochthonous placers.

Development Stages of Fluvial Land Forms and Varieties of Placers

Terms generally applied previously to alluvial placers connect them with various fluvial land forms and, accordingly, with the development stages of these forms. As the new division of alluvial placers into subtypes (autochthonous and allochthonous) and groups (bottom and above-bottom) is virtually independent of land forms, special attention must be given to the coordination between these new divisions and the older terms.

Channel and valley placers were first distinguished by Y. A. Bilibin, who indicated that channel placers, described as "the placers in a process of rebuilding," occur at the contacts with river channels and become valley placers during the filling of the valleys with alluvium of normal thickness, which break off the contact between placer and channel. Now, we can say that channel placers exist during river downcutting and turn into valley placers with the transition of the river to the stage of equilibrium. As above-bottom and allochthonous placers cannot form during downcutting, the channel variety can occur only among bottom placers.

Above-bottom placers may also occur in contact with river channels and then migrate from this contact. But such a change does not necessarily reflect a substantial turning point in the development of river valleys and therefore of above-bottom placers. Moreover, meandering rivers in equilibrium stage may repeatedly contact or diverge from above-bottom placers without any essential modification of their peculiarities. Thus, there are no reasons for dividing above-bottom placers connected with modern flood plains into channel and valley varieties. They represent a single variety so the term "valley placers," which is accepted for bottom placers connected with the same land forms, should be used for this variety of above-bottom placers.

Allochthonous placers formed during equilibrium stages are point-bar placers, and allochthonous placers connected with aggradational (accumulative) land forms comprise delta placers (placers of alluvial fans may also be included) and river-plain placers. Allochthonous placers may also be formed by aggrading rivers which do not create new land forms, but only transform their perstrative, or occasionally instrative, flood plains into constrative ones. However, such placers were not described until 1968, when looking through prospecting data on the Kuranakh gold placer (Aldan district) I concluded that the

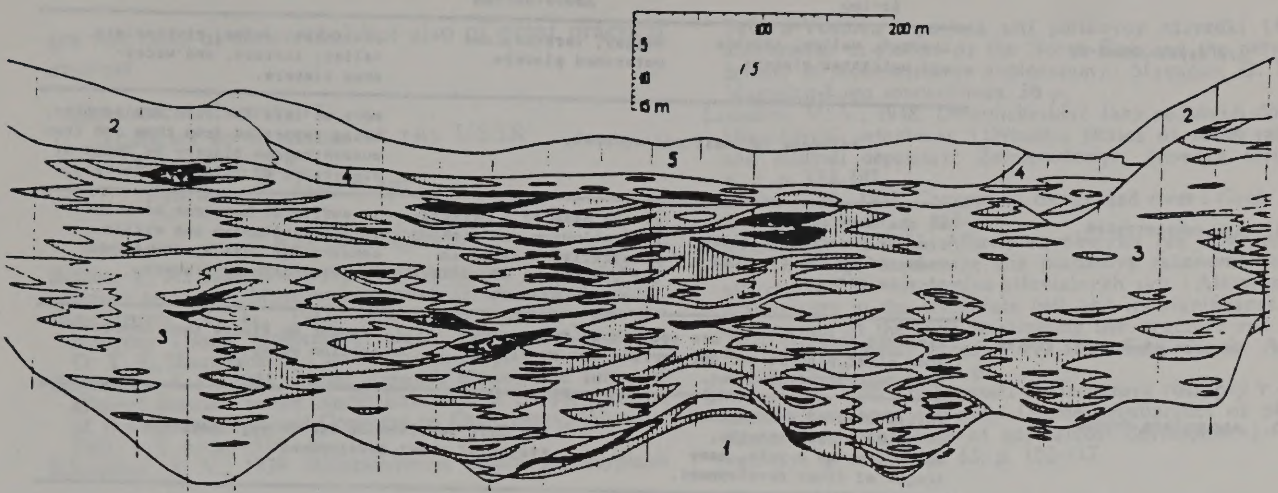


Fig. 4. An allochthonous gold placer of the Kuranakh River (a cross-section on a prospecting line). Thin shading, dense shading, and solid black designate an increasing content of gold. 1—bedrock; 2—colluvium; 3—constrative and underlying perstrative and substrative alluvium; 4—perstrative alluvium of the last equilibrium stage; 5—dredge dump.

main part of this valley placer concentrated in a constrative alluvium of a flood plain of the Kuranakh River is of allochthonous origin (Fig. 4).

Two sorts of gold may be clearly distinguished in this placer; gold from local ore sources is represented by coarser grains, and gold from sources occurring some 25 kilometers away consists of finer grains. The small nonpersistent concentrations at the bedrock are, apparently, derived from local ore sources and have to be regarded as bottom placers, and probably, there are also above-bottom autochthonous placers. However, as the fine grains brought by rivers from remote ore sources comprise most of the Kuranakh gold, most of the concentration have an allochthonous origin. Hence the valley placers of the Kuranakh River consists of autochthonous bottom and above-bottom placers as well as of allochthonous placers. Thus, the term "valley placer" can be applied to placers of the two subtypes and two groups of an autochthonous subtype, and it clearly should not be used without additional explanations.

Varieties of placers connected with land forms representing the next stages of development of flood plains also require explanation. The main types are terrace and watershed placers. The origin and morphological features of normal terrace placers of different subtypes and groups seem obvious. Relic land forms other than terraces are formed by changes in drainage pattern resulting from river beheadings, glacial advances, and other phenomena that deprive some valleys, or parts of valleys, of their rivers. Such abandoned or "dead" valleys may become parts of watersheds subject to processes of slope denudation, and under the influence of these processes they gradually lose morphological features indicating their fluvial origin. At the final stages of development, it is impossible to distinguish those relic fluvial land forms from other parts of watersheds. However, their alluvium may locally contain placers of different subtypes and groups that has escaped destruction. Such relic mineral concentrations are called watershed placers.

A summary of the geological characteristics of autochthonous (bottom and above-bottom) and allochthonous placers, showing differences between these subtypes and groups of placers and listing the varieties of placers characteristic for each, is given in Table 1.

Autochthonous and Allochthonous Placers of Various Minerals

The same "placer" minerals can form both autochthonous and allochthonous placers, however, every mineral "prefers" a certain subtype of placer. Some of them do not demonstrate this "preference" too clearly. For example, autochthonous gold placers, predominate over allochthonous ones, but the latter, particularly the point-bar variety are not uncommon. Alluvial placers of some other minerals, such as zircon, seem to belong to an allochthonous subtype alone.

Evidently this "preference" for certain subtypes of placers, as well as the extent to which it is pronounced, depends on the sum of such properties of minerals as density, predominant grain size, resistance to attrition and chemical weathering, fragility, etc. Each of these properties influences the behavior of minerals in flowing water. For example, the boundaries between "passive" and "active" fractions are determined mainly by density of minerals.

Table 1

Characteristics of Kinds of Alluvial Placers

	Autochthonous		Allochthonous
	Bottom	Above-bottom	
1. Are represented by	channel, valley, terrace and watershed placers.	valley, terrace, and watershed placers.	point-bar, delta, river-plain, valley, terrace, and watershed placers.
2. Occur	adjacent to their ore sources.		more or less far from ore sources, being separated from them and from autochthonous placers by zones of dispersion of "placer" minerals.
3. Are concentrated	at the base of an instrative or substrative alluvium and in the crevices of a bedrock.	at the base of a perstrative alluvium and within constrative strata, in the same parts of valleys as bottom placers.	in surficial horizons of a perstrative alluvium and within constrative strata, downstream of autochthonous placers.
4. Consist of mineral grains	received directly from ore sources or redeposited from older placers and not carried out by rivers from concentration zones.		brought by rivers into concentration zones.
5. Accumulate during	entire time of destruction of primary ore deposits, embracing, as a rule, many stages of river development.	the last equilibrium and/or aggradation stages of river development.	
6. An enclosing alluvium is formed during	downcutting stages or transition from them to equilibrium stages.	equilibrium and/or aggradation stages.	
7. An enclosing alluvium being rewashed during a downcutting stage, are	not destructed but displaced to the level of a new bedrock bottom.	displaced to the level of a new bedrock bottom and added to bottom placers.	completely destroyed.
8. The mechanism of concentration of "Placer" minerals	does not essentially depend upon hydrodynamic properties of flowing water.		depends to a great extent upon hydrodynamic properties of flowing water.

Heavier minerals have a greater number of grains forming autochthonous placers relative to the quantity of grains carried out of the site of concentration. This seems to explain the paradoxical fact that heavier minerals, as a rule, form autochthonous placers of greater extent than lighter minerals, as for example, in the autochthonous placers of gold and cassiterite.

However, none of the properties of minerals by itself has a *decisive* influence on their capability to form placers of either subtype. This is true even for density, as both the heaviest (gold and platinum) and lightest (diamond) "placer" minerals form placers of both subtypes. The gold and diamond placers demonstrate, too, that it is true for resistance of minerals to attrition. Similar examples could be quoted for any property of "placer" minerals.

Most of the mentioned examples refer to gold placers which are best known. Nevertheless, all the data available on placers of other minerals belonging to different subtypes and groups indicate that the main regularities of formation are the same for placers of all minerals. The principal differences between the placers of different minerals, probably, can be accounted for by these placers belonging to different subtypes or groups. There is no doubt that placers of different minerals belonging to the same subtype and group may differ in their length, thickness, content of minerals, persistence of this content, and so on, but such features may be different even in placers of the same mineral. The main geological features listed in the above table are the same in placers of every mineral.

The division into autochthonous and allochthonous subtypes applies not only to alluvial placers, but similar subtypes can also be clearly distinguished among coastal placers of lacustrine or marine origin. Most of these placers consist of mineral grains brought down by rivers and later displaced by wave action and other coastal processes. However, some coastal placers are formed at the expense of wave-cut primary ore deposits and are composed of mineral grains which are little displaced by coastal processes.

In conclusion, I would like to emphasize that the differences between the bottom, above-bottom, and allochthonous placers lead to the necessity for quite different methods in seeking, prospecting, and sampling them. Hence, the division of any type of placers into subtypes and groups, and an investigation of characteristic features of every subtype and group

are not only of theoretical but also of great practical interest.

Geological Institute,
Academy of Sciences of the USSR
Moscow, USSR,
February 9, April 7, 1971

REFERENCES

Bilibin, Y. A., 1938, Osnovy geologii rossypey (Principles of placer geology): Moscow-Leningrad, G. O. N. T. I., 505 p.

Bondarenko, N. G., 1957, Nekotoryie voprosy geologii rossypey (Some problems of placer geology): Magadan, O. T. I. Magadanskogo sovnarkhoza, 58 p.

Kartashov, I. P., 1965, Facies, dynamic phases, and suites of alluvial deposits in the North-East of the USSR: Report of the 6th International Congress on Quaternary, Warsaw, 1961, v. 1, p. 567–574.

Khripkov, A. V., 1958, Raspredeleniye zolota v rossypiakh Severo-Vostoka i gustota seti poiskovoy razvedki (Gold distribution in placers of the North-East and the network density in reconnaissance exploration): Magadan, O. T. I. Magadanskogo sovnarkhoza, 56 p.

Lamakin, V. V., 1948, Dinamicheskiye fazy rechnykh dolin i alluvialnykh otlozheniy (Dynamic phases of river valleys and alluvial deposits): Zemlevedeniye, (novaya seriya), v. 2, p. 154–187.

Mackin, J. H., 1948, Concept of the graded river: Geol. Soc. Am. Bull., v. 59, p. 463–511.

Shantser, E. V., 1951, Alluviy ravninnykh rek umerennogo poyasa i yego znacheniye dlia poznaniya zakonomernostey stroyeniya i formirovaniya alluvialnykh svit (Alluvium of plain-rivers in the temperate belt and its significance for recognition of the laws determining the structure and formation of alluvial suites): Tr. Inst. Geol. Nauk. Akad. Nauk SSSR, =135, 274 p.

Shilo, N. A., 1956. Osobennosti obrazovaniya rossypey v zone razvitiya vechnoy merzloty (Some peculiarities of placer formation in the zone of permafrost development): Sovetskaya geologiya, v. 53, p. 102–117.

THIS PAGE INTENTIONALLY BLANK

Economic Geology
Vol. 72, 1977, pp. 691-697

SCIENTIFIC COMMUNICATIONS

STREAM JUNCTIONS—A PROBABLE LOCATION FOR BEDROCK PLACERS

Introduction

A difficulty with the location of placers in a valley or in an alluvial deposit is, as stated by Crampton (1937), that "each placer deposit, whether modern or ancient, exposed or buried, affords its individual and peculiar problem." Therefore, there may be preferred positions of placers in alluvial fills and on bedrock valley floors, but an understanding of the channel history and the factors controlling deposition of the fill in each reach of a valley must be obtained before predictions of placer locations can be made.

Important concentrations of heavy minerals occur on the floor of valleys, on or near bedrock (Yeend, 1974). In fact, Cheney and Patton (1967) state that the concentration of heavy minerals within or near bedrock or a false bedrock is axiomatic. They suggest that this is due to the reworking of the alluvium during infrequent floods of unusually large magnitude, and they cite evidence from stream-gaging stations that indicate considerable scour during large floods. This, of course, is the idea that Dury (1964) and Palmquist (1975) advanced concerning the origin of the deepest part of a valley floor. However, no single explanation is adequate to explain all concentrations of heavy minerals on the bedrock floors. For example, Gunn (1968) suggested that agitation of sediments during flow permits downward movement of the heavy minerals, and Tuck (1968) believed that the explanation was that rivers concentrate heavy minerals on bedrock during periods of downcutting, and that these concentrations are then buried during aggradation. Of course, during and following changes of base level, there is scour and redeposition of sediments, so through time any change of base level will cause reworking of gravels and heavy mineral concentration. These concentrations often are related to positions of maximum scour on the valley floor.

During experimental studies (Shepherd and Schumm, 1974), when incision of a channel into resistant materials (mixture of kaolinite and sand) had reached maximum depth, aggradation was induced either by increasing sand feed at the head of the flume or by raising base level. In the first case, a wave of sediment moved progressively downstream so that deposition occurred first in the upper parts of the channel. The gradient was steepened there, and sediment moved progressively downstream. In the second case, deposition occurred at the lower end of the flume, and the channel was backfilled. In both cases a sediment containing 0.38 percent magnetite was moving through the flume, and the coarsest particles plus the magnetite formed a lag on the channel floor. As aggradation progressed, the magnetite concentration was buried and preserved on the floor of the valley. Above this bedrock concentration a valley-fill deposit that had not been reworked contained magnetite but not in noticeable concentrations. Deposition was not uniform over the bedrock floor of the channel, but magnetite concentrations occurred on both low and high points of the bedrock floor. In pools the heavy material was trapped and concentrated with the coarsest material, but where the valley floor was high, channel width was also greatest and flow velocities were locally reduced, thereby causing deposition of the coarser and heaviest sediment at these locations also. It appears, therefore, that heavy minerals can always be expected on the bedrock floor of valleys if the river was transporting heavy minerals during erosion of the valley floor and deposition of the overlying sediments.

The experimental study and field evidence (Shepherd and Schumm, 1974, figs. 3 and 6) indicates that the bedrock valley floor will be irregular both in cross and longitudinal profile and that especially favorable sites of placer formation are related to bedrock scour, which can take the form of continuous bedrock channels or discontinuous but regular patterns of scour. Although such patterns exist, the prediction of their location in a valley is difficult; however, another experimental study which was performed as part of an investigation of the influence of tributaries on main channel morphology (Mosley, 1976) produced information that may be relevant to this problem.

Experimental Study

Procedure

These experiments were conducted in a flume that is 2.8 m long and 1.3 m wide, with a longitudinal slope of 0.016. The flume was filled with a cohesive sediment with a median diameter of 0.34 mm and a

FIG. 1. Scour hole in a typical model confluence, looking upstream. Note the concentration of magnetite (dark band in center of channel) downstream from the confluence.

silt-clay content of 19 percent which had been passed through a 2-mm screen to remove pebbles and clay lumps. Water was supplied to the head of the flume via two independently controlled supply lines; the combined discharge was always 250 cc/sec. Ninety-five experimental runs were carried out; tributary confluence angle θ, the ratio of tributary discharges Q_1/Q_2, tributary widths w_1 and w_2, and tributary sediment loads QS_1 and QS_2 were varied to evaluate their effects upon maximum water depth D_s in the confluence. Four runs in straight unbranched channels flowing at discharges of 250 cc/sec were also made for comparison. They provided a value for mean channel depth $\overline{D}$ of 0.78 cm to which the depth of scour (Ds) at tributary junctions may be compared.

Runs were terminated when the tributaries and confluence had reached an equilibrium state in which all sediment fed at the head of the tributaries was transported through the flume, and significant erosion and deposition had ceased. Longitudinal profiles and selected cross sections were measured to enable computation of values for stream power, width-depth ratio, and other variables describing the hydraulic and morphologic character of the channels. Cross sections and maximum water depths in the confluences were also measured, and dye was introduced into the flow to aid in observation of flow patterns at the confluence.

Experimental Results

As sediment was fed at the head of the tributary channels, sand, traveling as bed load, formed a dune front in each channel, which moved downstream. When the dunes reached the confluence, they rotated to parallel the flow in the lower channel, forming a scour hole which was, therefore, partly depositional in nature (Fig. 1). Bed erosion at the location of the scour hole indicated that the flow patterns that maintained the feature existed independently of it. Dye injection demonstrated that flow patterns in the confluence were very complex. Shear along the boundary between the two converging flows at the confluence set up vertical vortices, which were swept downstream. They were evidenced by the distinct but scalloped boundary between the two merging tributary flows.

Flow from the tributaries crossed to the center of the confluence at the surface, plunged to the bottom, then returned to the surface along the walls of the scour. Two opposed helicoidal cells were thus established in the scour hole: two full cycles of each helix were visible. The helicoidal cells prevented movement of sediment into the center of the scour holes, so that the sediment entering from the tributaries was transported along the base of the scour walls and left it in two lateral zones. These converged some distance downstream from the confluence: in between, a band of magnetite was usually found (Fig. 1).

The scour holes were evidently a result of the shear and turbulence generated as the confluent flows were turned into the continuing channel. Description of the precise nature of turbulence in such a complex situation is virtually impossible, but the degree of turbulence and rate of energy dissipation are related to the change in flow momentum in the confluence, that is, to the forces exerted on the flows as they pass through the confluence (Chow, 1959).

The momentum equation shows that these are in turn related to confluence angle, water discharge, water density (constant in the experiment), and flow velocity (approximately 0.3 m/sec in all runs). Figures 2 and 3 show relative scour depth $D_s/\bar{D}$ as a function of tributary confluence angle and the ratio Q_1/Q_2 for two subsets of the 95 runs. Runs with straight unbranched channels and discharges of 250 cc/sec are also included. At confluence angles of 15° there was little shear along the boundary of the converging flows, and hence little tendency for a scour hole to form. This corresponds to engineering experience with generation of turbulence in rigid boundary confluences (Corps of Engineers, 1970). Qualitative observation indicated that shear and turbulence in the scour increased rapidly above 15° to angles of 90°, and more slowly thereafter; scour hole depths show a similar variation with confluence angle (Fig. 2).

Figure 3 shows that, as the smaller tributary (with discharge Q_1) increases in discharge relative to the larger, bed scour in the confluence becomes more pronounced. When all flow is in one channel, shear and turbulence are at a minimum, and water depth in the confluence is equal to the depth in a straight channel. As flow in the smaller tributary is increased, it exerts an increasingly large turning force on flow in the larger, shear and turbulence are generated and helicoidal flow cells become established in the confluence, and bed scour increases. The net amount of turning of the flow, generation of turbulence, and hence depth of scour are greatest when the tributary has from half the discharge of the main channel to an equal discharge. Tributary widths were varied in the subset of runs shown in Figure 3 but had no clear effect on scour depths.

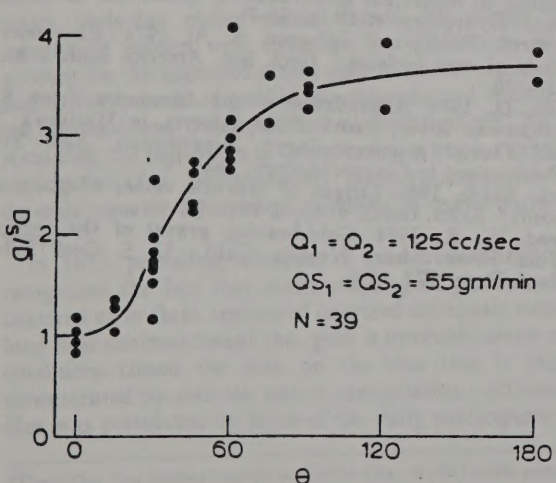


FIG. 2. Relation between relative scour hole depth ($D_s/\bar{D}$) in the confluence and confluence angle θ. Mean depths for four straight channels ($\theta = 0°$) are included.

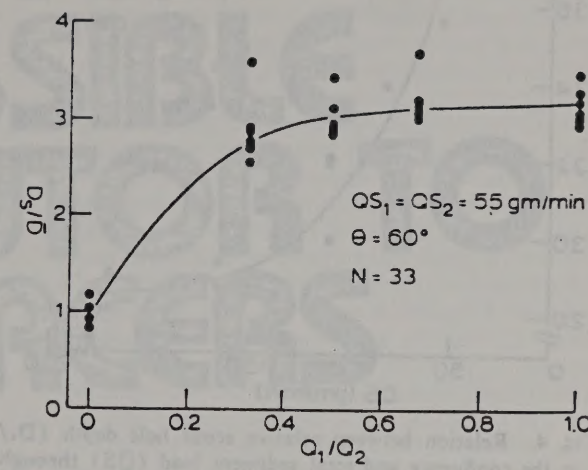


FIG. 3. Relation between relative scour hole depth ($D_s/\bar{D}$) in the confluence and tributary discharge ratio (Q_1/Q_2). The three outliers were for runs in which flow in the tributaries was not uniform across the channel but became concentrated near one bank. Discharges per unit width were therefore unusually high and caused increased scour.

Figure 4 shows the effect of varying sediment load, with tributary discharges constant. As sediment load increases, relative scour depth decreases; varying relative tributary sediment load QS_1/QS_2 had no discernible effect upon D_s. Apparently an increase in sediment transport requires an increase in bed shear and flow velocity, which implies constriction of the flow cross section and a decrease in flow depth.

Observation of branch channel confluences in Medano Creek and the North Platte River, braided streams in Colorado and Nebraska, provided qualitative field evidence for scour features like those observed in the flume. Smith (1974) has also observed such features, up to 3 m deep, in the North Saskatchewan River. A small scour 25 cm deeper than general bed level was observed in the confluence of Red Dirt Creek and Big Muddy Creek, Colorado. The boundary between the confluent flows was scalloped, as in the flume, and vertical vortices which were swept downstream were clearly visible. Bankfull depth in the confluence was 1.7 m, but flow was 0.8 m below bankfull at the time of the visit, which probably explains the small dimensions of the scour. A large scour hole formed during channel-forming discharges would probably be filled in as discharges declined, and evidence for its existence would therefore be lost. In addition, flood peaks from the two tributaries are probably rarely coincident, so that the depth and orientation of a scour hole in a confluence would be expected to vary with the passage of flood waves through the channel.

Several studies have shown that channel beds are scoured during flood events if alluvium is of moderate thickness (Pickup and Warner, 1976). Palmquist

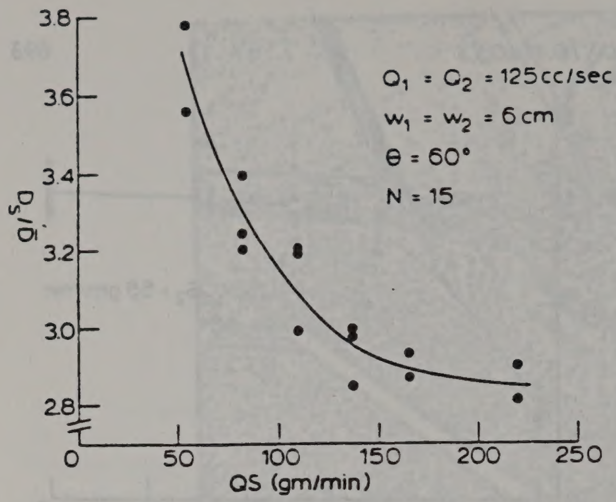


FIG. 4. Relation between relative scour hole depth ($D_s/\bar{D}$) in the confluence and total sediment load (QS) through the confluence. Note that confluence angle θ was 60° throughout: Figure 2 suggests values of $D_s/\bar{D}$ of about 3.0 when $\theta = 60°$ and QS = 110 gm/min.

(1975) has suggested that this periodic scour may, over a long period, cause erosion of the bedrock valley floor to depths of two or three times normal bankfull depth. Because streams tend to be located at preferred positions within a valley, such as on the outside of valley bends, depths of bedrock erosion should tend to vary and to be in proportion to the length of time that a stream is located at a given point. It appears that scour at channel and valley confluences may be integrated into Palmquist's "preferred position model". Periodic scour at confluences should, over the long term, cause erosion of the bedrock floor to greater depths than elsewhere in the valley because of the greater turbulence and erosivity at the confluence of two flows.

Summary

The results from the experimental study of stream confluences indicate that bedrock scour depths should be greatest at the confluence of streams. The scour will be deepest at junctions of tributaries that have discharges (drainage areas) that are from half to equal the amount conveyed by the main channel (Fig. 3), have small bed-load transport rates (Fig. 4), and meet at angles between 60° and 90° (Fig. 2). Field observations are not available to test these conclusions; nevertheless, they may have important implications for the location of placer deposits. Needless to say, the enhanced sediment reworking due to the turbulence at tributary junctions will promote heavy-mineral concentrations at these locations.

As placer minerals are known to be deposited preferentially in and around depressions in the bedrock surface, valley confluences would seem to be very promising areas for prospecting. It is significant in this respect that segregation of magnetite occurred just downstream of the scour holes in the model channel confluences (Fig. 1). In addition, if there is a repeating pattern of bedrock scour on valley floors (Shepherd and Schumm, 1974, figs. 6 and 7), the uppermost scour of the series will be at a tributary junction.

Acknowledgments

We thank Professor T. B. Thompson for his review and the National Science Foundation for financial support under Grant GA 38606.

M. P. MOSLEY
S. A. SCHUMM

DEPARTMENT OF EARTH RESOURCES
COLORADO STATE UNIVERSITY
FORT COLLINS, COLORADO 80523
May 17, 1976

REFERENCES

Cheney, E. S., and Patton, T. C., 1967, Origin of bedrock values of placer deposits: ECON. GEOL., v. 62, p. 852–860.
Chow, V. T., 1959, Open channel hydraulics: New York, McGraw-Hill, 680 p.
Corps of Engineers, 1970, Hydraulic design of flood control channels: Engineer Design Manual EM-1110-2-1601.
Crampton, F. A., 1937, Occurrence of gold in stream placers: Mining Jour., v. 20, p. 3–4, 33–34.
Dury, G. H., 1964, Principles of underfit streams: U. S. Geol. Survey Prof. Paper 452-A, 67 p.
Gunn, C. B., 1968, Origin of the bedrock values of placer deposits: ECON. GEOL., v. 63, p. 86.
Mosley, M. P., 1976, An experimental study of channel confluences: Jour. Geology, v. 84, p. 535–562.
Palmquist, R. C., 1975, Preferred position model and subsurface symmetry of valleys: Geol. Soc. America Bull., v. 86, p. 1392–1398.
Pickup, G., and Warner, R. F., 1976, Effects of hydrologic regime on magnitude and frequency of dominant discharge: Jour. Hydrology, v. 29, p. 51–76.
Shepherd, R. G., and Schumm, S. A., 1974, Experimental study of river incision: Geol. Soc. America Bull., v. 85, p. 257–268.
Smith, D., 1974, Aggradation of the Alexandra-North Saskatchewan River, Banff Park, Alberta, *in* Morisawa, M., ed., Fluvial geomorphology: Binghamton, New York, Publications in Geomorphology, chap. 9.
Tuck, Ralph, 1968, Origin of bedrock values of placer deposits: ECON. GEOL., v. 63, p. 191–193.
Yeend, W. E., 1974, Gold-bearing gravel of the ancestral Yuba River, Sierra Nevada, Calif.: U. S. Geol. Survey Prof. Paper 772, 44 p.

LATERIZATION AS A POSSIBLE CONTRIBUTOR TO GOLD PLACERS

laterization - converts rock to soil
residual prod. of weathering

David Le Count Evans, Consulting geologist*

Twenty four years of observation have gradually suggested to the author that gold, in small amounts but evenly distributed throughout ultrabasic rocks, may be chemically dissolved and re-precipitated in possible commercial placer concentrations by the normal process of laterization.

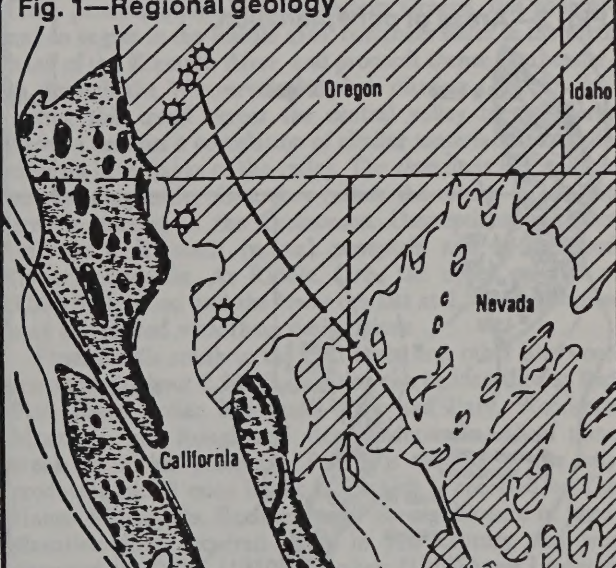
The arrival at such a premise has been accompanied by some troubled thinking dating back to 1955, after a detailed study of a French Guiana gold placer. At that time, it was concluded that the only possible source of the placer gold was the thick, red, lateritic soil cut by the stream channel. Of particular interest was the fact that some of the gold in the channel obviously had been re-precipitated from solution.

If this premise is true, it may account for some of the gold deposits in the higher Tertiary channels of California's Mother Lode country and on the west slope of the Sierra Nevada. Furthermore, this premise may explain the occurrence of extremely large nuggets of exceptional fineness, which early-day placer miners would not accept as the product of eroded vein structure. It might also partially account for the scattered placer gold in southwestern Oregon and the abundance of gold in the channels and tributaries of the Klamath and Trinity Rivers of northwestern California. Assuming the soundness of this premise, a key is suggested to exploration for undiscovered, large-yardage, low-grade deposits minable by open-pit methods in many parts of the world.

In 1955, prevailing concepts of gold mineralization only recognized the fact that many major gold placer areas are centered on or flank regions of laterized ultrabasic rocks. The long-held commandment that gold is insoluble under normal conditions closed the door on the idea that it might be concentrated by solution and re-precipitation—although this idea was postulated by some of the early participants in the California gold rush. By 1972, however, contributions to the literature were providing the chemistry needed to permit questioning of gold's presumed insolubility.

Such a suggestion does not imply that all lateritic deposits contain unrecognized secondary concentrations of gold. Much less does it suggest that the many successful nickel-silicate mines, as a group, have been overlooking by-product

Fig. 1—Regional geology



*The author first studied laterites at Riddle, Ore., in 1941 while participating in the discovery and exploration of the Eight Dollar Mountain deposits. In 1942 and 1952, he examined the main nickeliferous laterites that continue south through California's Del Norte County. He has also worked in French Guiana, the California Mother Lode area, Siskiyou and Trinity counties in California, and Costa Rica.

Secondary origin of Fe, AP,

gold values. In fact, study of many excellent papers on worldwide nickel operations has revealed no references to gold—not in traces measurable in parts per million, not in raw ore or concentrates, and not in the refined product. The suggestion does permit consideration of the possibility that nickel-cobalt and gold deposits separately but not in combination may have a common lateritic origin.

Lateritic deposits result from the alteration of rock masses. The conditions that produce such alteration include a tropical climate, to promote steady chemical disintegration; a regionally flat surface and similarly flat water table, to assure little movement of solutions after they enter the zone of saturation; and long time intervals, to allow the chemical action to penetrate great thicknesses of rock. Laterization of acid, light-colored rocks has produced bauxite deposits in the Caribbean, along the northern coast of South America, in the southeastern US, and in Costa Rica and Australia, among other locations. Laterization of peridotites, dunites, gabbros, and other ultrabasic rocks has fostered the nickel silicate deposits of New Caledonia, Cuba, the Philippines, Colombia, Guatemala, Venezuela, Brazil, Oregon, northwestern California, and other regions.

ENIGMA OF CALIFORNIA GOLD

From the earliest days of the California gold rush, San Francisco journals indicated evidence of interest in the actual origin of the state's placer gold, especially in the drift mines that followed old Tertiary channels. THE MINING AND SCIENTIFIC PRESS carried letters from miners and discussions by leaders in the scientific community that questioned accepted theories. An article by a Dr. Landsweert (1869) is representative of the general tone of this debate. He wrote: "Every one concurs in the belief that alluvial gold has been derived at some time or other from lodes; but seeing that the largest piece of gold ever found in the matrix is insignificant when compared with the nuggets that have sometimes been found in the alluvium, it has been a difficult matter to reconcile belief with experience."

Dr. Landseert described a number of experiments in which very dilute solutions of gold chloride and a carbonaceous material used as a "decomposing" agent were mixed to precipitate films of gold on centers of pyrite, iron, copper, antimony, galena, and other metals and minerals. Using brown iron ore, gold was deposited as a fine metallic powder.

Dr. Landsweert concluded: "The occurrence of larger nuggets in gravel deposits than have been found in quartz ledges, with the fact that alluvial gold almost universally has a higher standard of fineness, would seem to imply a different origin for the two." He also quoted a contemporary, a Professor Bischoff, who added fuel to the fire by saying: ". . . the very marked difference between much of the alluvial gold and vein gold indicates that the alluvial had some other origin and was transferred from the reefs to the drifts by some means other than denudation; and when we consider that the nuggety gold consists of nearly the heaviest known matter, offering but a small surface of attack when compared to the other constituents of the veins, it appears strange that they should be found at a great distance from any known reefs, as nearly all the large nuggets have been."

Such discussion naturally drew opposition from those who defended the gospel of denudation and mechanical transport of gold from source rock.

In further support of the solution hypothesis, Professor Joseph Le Conte (1871) offered the following: "Will sulphate of iron dissolve gold? We must remember in this connection the difference between the operations of the chemist and those of nature. The chemist calls a substance insoluble if it

Fig. 2—Areas of peneplanation

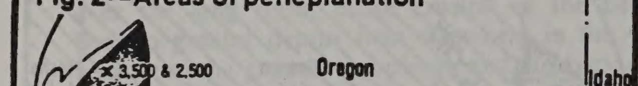
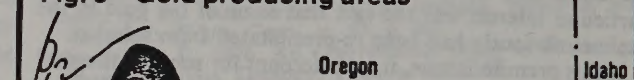


Fig. 3—Gold producing areas



is not dissolved to any considerable extent in a limited space of time. But nature has infinite ages of geologic time to work. Hence, a substance dissolved only in part by the chemist may be entirely dissolved by nature. . . ."

Describing the pocket deposits of eastern Trinity county and western Shasta county, California, O. M. Hershey (1910), observed: "The gold lies in a thin, flat sheet . . . it is in the form of coarse and fine grains that have a peculiar smooth and rounded surface, quite *unlike* the fine gold in quartz veins . . . I believe that some of the finely divided gold (detritus from base-metal veins with very low gold values) passes into solution in the zone of oxidation and is leached out by the descending meteoric waters . . . and some may be deposited in cracks in the rock to form the so-called 'pocket' seams.

"This certainly is secondary concentration by meteoric waters, circulating near the surface, though it is not secondary enrichment in the sense ordinarily meant."

Hershey remained undecided as to whether deposition occurs in the zone of oxidation or just below it, but he concluded: ". . . (the gold) is transported in solution in ordinary cold meteoric waters, circulating within several hundred feet of the surface. . . ."

F. W. Clarke (1908) pointed out that the natural solvents of gold appear to be numerous. Gold is held in solution by potassium silicate, and it is perceptibly soluble in a 10% sodium carbonate solution or a mixture of sodium silicate and bicarbonate. Solutions of alkaline sulphides have been found to be effective solvents. Hydrogen sulphide attacks gold perceptibly. *All* of these solvents occur in natural waters.

Clarke referred to an experiment by T. A. Rickard in which a sample of Cripple Creek ore, containing manganic oxides, was treated with a solution of ferric sulphate, sodium chloride, and a little sulphuric acid. All of the gold was dissolved. A fragment of black carbonaceous shale was immersed in the solution, and the gold was precipitated.

The experiment illustrated the ease with which gold is redeposited from its solutions. Organic matter of almost any kind will precipitate gold, and such matter is rarely, if ever, absent from the soil. Gold, therefore, although it may enter into solution, is not likely to be carried far before it is precipitated. Gold is thrown out of solution by ferrous salts, other metals, and many sulphides, as well as ordinary soil.

RECENT LABORATORY EFFORTS

Recent studies, although not designed to support the premise of laterization as a source contributing to gold in placers, *do* support the solubility of gold in nature. Cloke and Kelly (1964) confirmed earlier work by Emmons (1917) and Krauskoph (1951) that indicated that acid chloride solutions in the presence of a strong oxidizing agent produce conditions favorable for the solution of gold. A solution of H_2SO_4 and NaCl exposed to pyrolusite and goethite on a gold plate, respectively, dissolved gold in appreciable amounts in both cases. Cloke and Kelly emphasized the fact that the only inorganic acid commonly present in the supergene environment is H_2SO_4, which is formed through the oxidation of sulphides remaining in the gossan.

Concerning supergene transport, Coke and Kelly reported that "the actual process of movement of gold is envisioned to involve repeated solution and deposition," and they concluded, "The preceeding evidence supports the conclusion that acid chloride solutions, together with iron and manganese oxides can provide supergene transport of gold."

With reference to goethite, Hotz (1964) listed limonite, goethite, and maghemite as constituents of the brick red surficial soil of Oregon's laterites, with these minerals characterized by shot-like granules. Maghemite is a magnetic anhydrous ferric oxide.

With reference to maghemite, Coveney (1972), reporting on the localization of high-grade gold ores in California's Oriental mine deduced that hydrogen gas accompanying magnetite at a serpentine contact was the reducing agent. Precise work indicated that the H_2 was not in the serpentine but in the bi-occurrences of magnetite.

Rucklidge (1972) concurs with White (1968) in the belief that chlorine is a significant element in environments where metals are being redistributed and deposited. He further states that high chlorine in serpentines comes from partial serpentinization. He concludes that "The influence of chlorine-bearing fluids on the movement and redistribution of metals from *ultramafic* rocks is a tantalizing subject."

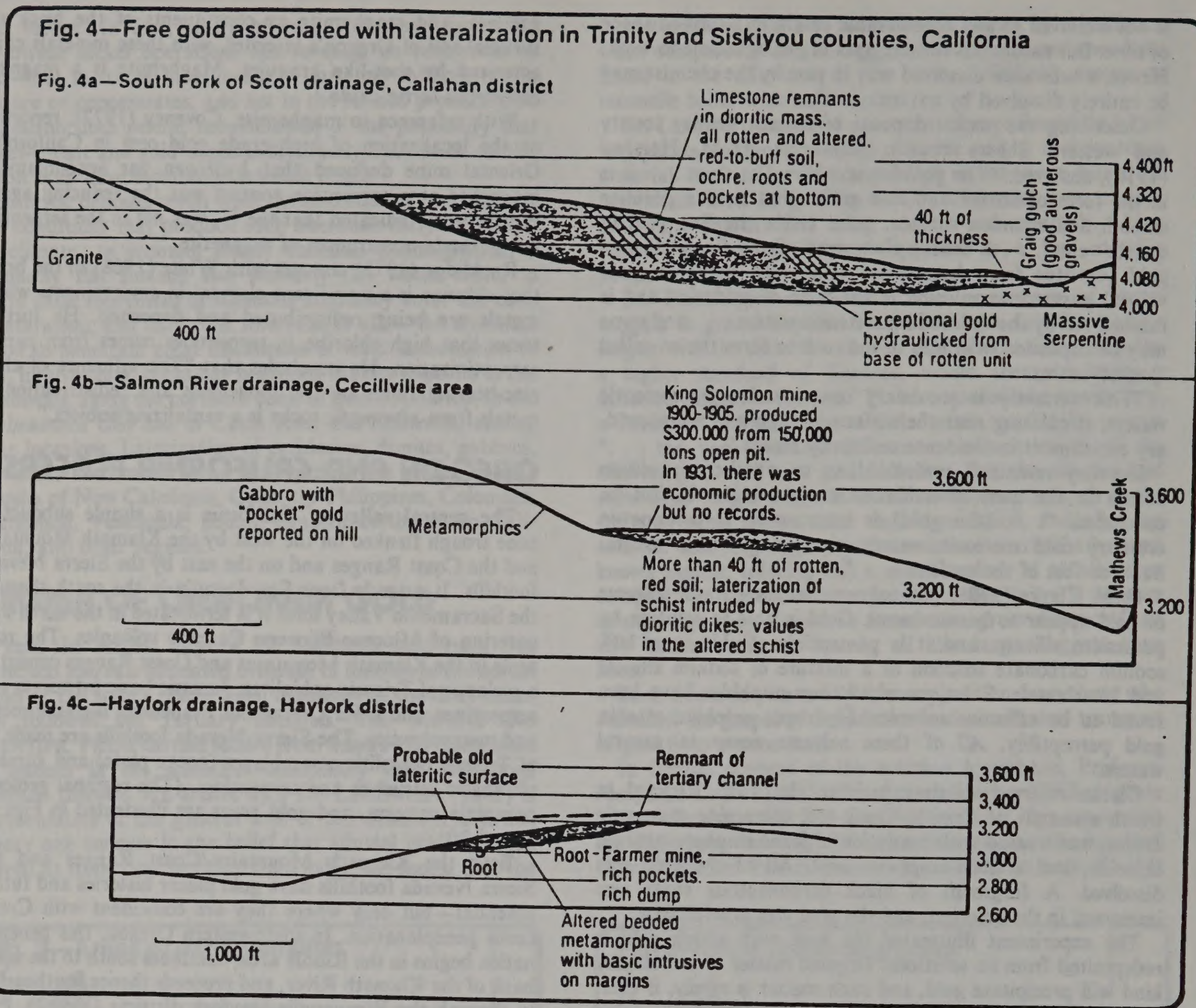
OREGON AND CALIFORNIA PLACERS

The central valley of California is a simple subduction zone trough flanked on the west by the Klamath Mountains and the Coast Ranges and on the east by the Sierra Nevada foothills. It extends from San Joaquin in the south through the Sacramento Valley until it is terminated in the north by a covering of Miocene-Pliocene Cascade volcanics. The rock units in the Klamath Mountains and Coast Ranges consist of a melange of Triassic ophiolites, Jurassic basic intrusives and serpentines, and lower Cretaceous Franciscan metasediments and metavolcanics. The Sierra Nevada foothills are made up of Triassic ophiolitic assemblages (ocean plate) and Jurassic ultrabasic intrusives and serpentines. (The regional geology, peneplain patterns, and gold areas are illustrated in Figs. 1, 2, and 3.)

Both the Klamath Mountains/Coast Ranges and the Sierra Nevada foothills have gold placer histories and future potential—but only where they are coincident with Cretaceous peneplanation. In southwestern Oregon, this peneplanation begins in the Riddle area, continues south to the north bank of the Klamath River, and proceeds thence southeasterly through the Weaverville-Hayfork districts (Fig. 2). Projecting this trend across the central valley to the Sierra Nevada permits a correlation of similar materials and conditions on both flanks of the valley. The elevations in Fig. 2 are not too diagnostic, since they reflect the results of tectonic growth starting in the Cretaceous. Descriptions by Diller refer to a Klamath (upper) peneplain and a Sherwood (lower) peneplain. At Riddle, Ore., the upper ore flat at 3,500 ft elevation and the lower ore flat at 2,500 ft elevation may correspond with these designations.

From Riddle south to the California line, other Sherwood remnants at about 2,500 ft elevation can be identified at Red Flat, Eight Dollar Mountain, Free and Easy, Woodcock Mountain, and Rough and Ready Mountain. All of these areas are nickel prospects. Riddle is the site of the only producing nickel mine in the US, which is operated by The Hanna Mining Co. Recognition of the significance of peneplanation first occurred early in this century and was discussed by Diller (1910), Hershey (1910), and Lindgren (1911).

Fig. 3 outlines gold-producing areas and illustrates their promising coincidence with peneplaned areas. In southern Oregon, the Rogue River and its tributaries have been worked extensively for placer gold, as have the Waldo-Jacksonville and Illinois River drainage areas south of the Rogue and the area south of Eight Dollar Mountain near Kirby. The Oregon State Geological Survey's Bulletin 61



provides good historical and geological detail about these districts. An association with an ultrabasic complex is suggestive.

SOUTHERN KALAMATH

Jurassic ultrabasics, laterized peneplain remnants, and exceptional placer gold deposits such as those found in southwestern Oregon also occur farther south in the Siskiyou-Trinity region of California. Peneplain remnants in this area include Pine Flat, 2,500 ft; Low Plateau, 2,600 ft; High Plateau, 3,500 ft; Little Rattlesnake, 3,200 ft; Rattlesnake, 3,600 ft; and Red Mountain, 4,000 ft, as well as four others that conform to requirements but have not been mapped. These four are in Del Norte county, north of the Klamath river and close to the coast.

Continuing southeasterly to the Weaverville area, other peneplain remnants include the King Solomon mine, 3,300 ft; the Crawford anomaly, 4,200 ft; and the Dog Paw laterite, 4,200 ft. There are also three such remnants in the Cecilville area on the Salmon River drainage and another at Wildcat Creek west of Callahan. All of these peneplain remnants are auriferous. The King Solomon and Wildcat Creek properties produced economically from rotten, red-to-buff, lateritic soil deposits. Neither reflects a vein source nor orthodox gravel characteristics.

Of special interest is section A of Fig. 4. The position of a thick layer of exceptional gold at serpentine bedrock and beneath barren (leached?), rotten, heavily oxidized diorite is considered very significant. Section B of the same figure shows laterized diorites and schists with some gold concentration at King Solomon's mine flanked by gabbro with scattered "pocket" gold. Section C shows the Hayfork district, with pockets or "roots" of gold beneath a remnant of stripped-away laterite.

Study of various records of gold production in the Klamath Mountain province in California indicates a total production of 11,369,000 oz, of which 87% can be attributed to placer mining.

MOTHER LODE AREA

The existence of peneplanation in the Sierra Nevada foothills was the subject of some debate in the early 1900s. Lindgren, in the classic "Tertiary Gravels of the Sierra Nevada of California" (1911), questioned such a thesis, attributing the regional flatness to level lava flows that conceal a pre-Tertiary surface. He qualified this view to some extent by noting that east of the depression between Forest Hill and North Columbia, the terrain rises to an

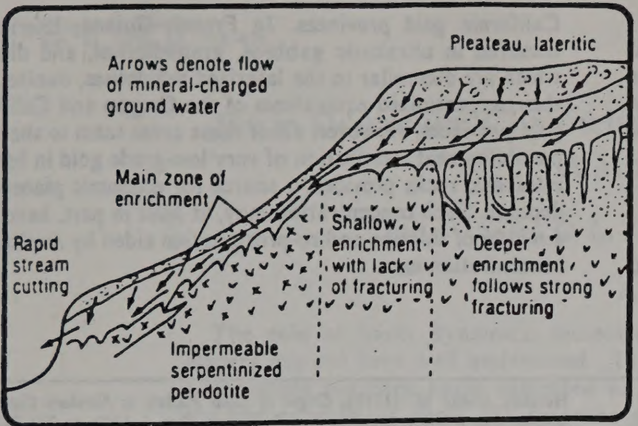


Fig 5—This composite shows how a lateritic nickel orebody can be enriched by migratory solution and indicates clearly the extent to which enrichment is guided by fractures in the underlying bedrock.

undulating plateau that extends over a considerable space at an elevation of 3,000 to 5,000 ft. (This area contains sufficient nickel silicate to have justified recent exploration.)

Others, including Diller (1910) and Hershey (1911), supported the peneplanation thesis, as do some researchers today. In a personal study, the author constructed profiles from Magalia to Forest Hill that indicates two terraces, at 4,300 and 3,300 ft. The width of the terraces is 15 mi, and the distance covered is 80 mi. There is also a suggestion of a third terrace at 2,500 ft. Direct observation in scattered areas reveals a surface of heavy red soil and an irregular contact between overlying laterization and underlying basic rocks, with some serpentinization. In many respects, the area is similar to laterized areas in southwestern Oregon and northwestern California.

Sierra Nevada placer gold, won from Tertiary channels and existing streams, accounted for about 66,817,000 oz of California's total gold production of 106,276,163 oz through 1970, 63% of the total. Further review of records indicates that these placers accounted for 74% of production in the Sierra Nevada province.

W. B. Clark (1979) notes that Eocene channels are the most extensive and productive in the Sierra and that they originated in Cretaceous time, when streams flowed across a subdued topography into a shallow Cretaceous sea west of today's Sierra. The Eocene climate is generally thought to have been sub-tropical, permitting deep weathering of rock.

These older Eocene channels consist of boulders of intrusive and metamorphic rocks, as well as white milky quartz. In some cases, there are "greenstone" gravels near bedrock that have "yielded significant quantities of gold." In all cases, the highest proportion of gold occurs at or near bedrock in the oldest deposits.

Gold particles range from flour size to coarse nuggets weighing more than 100 oz. Fineness of the gold is exceptional, ranging from 850 to 950.

Thus, the Sierra Nevada portion of the Oregon-Klamath-Sierra trend repeats certain basic conditions. The gold-producing areas occupy certain Triassic-Jurassic trends of ultrabasic intrusives but only where overlain by Cretaceous peneplanation that was followed by laterization of the ultrabasic masses. The fineness (purity) of gold is exceptionally high. Production in all cases was in part from well-developed vein structures in the Jurassic basic units and in part from placers, which represent 60-87% of an area's total production. The few isolated observations in the Klamath area of placer concentrations tied to laterization cannot be repeated for the Sierra, possibly because of insufficient work.

PRE-CAMBRIAN FRENCH GUIANA

In 1955, the author undertook studies of placer deposits in French Guiana, where, except for a thin strip of Quaternary shales and sands on the coast and some sandstones and conglomerates of Cretaceous age in a few streams, all of the rock is of pre-Cambrian age. An upper complex of schists and conglomerates is designated the Orapu series, and a lower complex of schists and quartzite is designated the Paramaka series. These rocks, with unclassified quartzites and amphibolites at the very base of the column, all rest on underlying granite. Before Orapu time, first diorites, granodiorites, and gabbros, and later granites invaded the Paramaka rock sequence.

Quartz veins that are slightly auriferous have been mined without economic success at St. Elie, Adieu Vat, and Bief. Veins also occur in the Sauvenier-Haute Mana area, on the Orapu and Comte Rivers, between the Orapu and Approuague Rivers, in the Arouany Basin, on Lezard Creek, and elsewhere. The gold is thought to derive from the diorites, granodiorites, and gabbros, which carry disseminations of pyrite and chalcopyrite and probably minor gold with the sulphides.

The French Guiana portion of the Guiana shield has been a positive area since at least pre-Cambrian time, exposed to a tropical environment on a flat terrain—the required conditions for laterization. There is evidence for at least two periods of vast peneplanation and "telescoping" of thick rock masses by chemical disintegration and reorganization. Brown ferruginous soil of varying thickness occurs as thin layers atop terrace remnants and covers the slopes away from the terraces. The ferruginous soils grade abruptly into massive secondary limonite and minor hematite at the contact with partially altered bedrock.

Where soils have been removed by erosion, massive iron oxide caps the upper elevations. Massive limonite with large voids in the matrix is known as "box-work" iron.

Streams flow through valley fill that consists of transported boulders of lateritic iron underlain by 5-15 ft of thick laterite-derived clay, which, in turn, caps several feet of gold-bearing sand and gravel.

Guianans had worked the few feet of auriferous gravels in current drainages, as well as so-called "hillside" placers that occur as detrital deposits on the flanks of streams well above drainage and in the laterite itself. These workers claimed they were getting better values, with less effort, from the "hillside" placers than from the present stream.

Supporting the possibility of gold being soluble in nature under the proper conditions, the author observed clusters of recent fine, clean sand grains wired together with fibrous, crystalline gold. Gold was also observed coating rounded pellets of iron oxide, in the voids of iron oxide, and as angular quartz pebbles. Fineness approached 950. No gold was noted in pure vein quartz.

Geologists working in French Guiana have noted that placer deposits occur where drainages cross areas of diorite, granodiorite, and gabbro. Choubert (pre-1955) states, "Our observations allow us to affirm that placers and veins (of gold) are concentrated exclusively in areas of dioritic intrusives, old lava flows, or both," and "gold-bearing laterites have been recognized in all of the Guianas."

Braceville (1946) reports, "Gold placers are found within or close to ridges of metamorphic rocks . . . they are more often than not situated in lateritic areas and the laterite itself often contains workable values . . . concentrations of the deposits by alluvial agencies has resulted in the production of extensive workable deposits in small gullies, streams, and on

hillsides.

"The bulk of the gold produced in British Guiana has probably been obtained from alluvial deposits not more than 6 ft deep, in which the gravel layer is usually 1 ft thick and contains values of about $2 per yard, or about 30¢ per yard overall value . . ."

SUPPORT FOR SOLUTION PREMISE?

There are differences between the French Guinana and California gold provinces. In French Guiana, laterization occurred in ultrabasic gabbros, granodiorites, and diorites, which are dissimilar to the laterized peridotites, dunites, and their serpentinized equivalents of the Oregon and California gold provinces. However, all of these areas seem to share the possibility that laterization of very low-grade gold in basic to ultrabasic rocks provided a source for economic placer gold deposits. Such concentration may, at least in part, have been a matter of solution and re-precipitation aided by mechanical concentration. ■

REFERENCES

Braceville, S. (1946), Geology and Mineral Resources of British Guiana.

Brooks, Howard C., and Ramp, Len (1968), *Gold and Silver in Oregon*, Bull. 61, Oregon Department of Geology and Mineral Resources.

Choubert, M. Boursi (pre 1955), Geologie et Petrographie de la Guyane Francaisie.

Clark, William B. (1970), *Gold Districts of California*, Bull. 193; California Division of Mines and Geology.

Clark, William B. (1979), *Fossil River Beds of the Sierra Nevada*, CALIFORNIA GEOLOGY, July 1979; pp 143-149.

Clarke, F.W. (1908), *Data of Geochemistry*, Bull. 330, US Geological Survey.

Cloke, Paul L. and Kelly, Wm. C. (1964), *Solubility of Gold Under Inorganic Supergene Conditions*, ECONOMIC GEOLOGY, Vol 59, pp 259-270.

Coveney, Raymond M. (1972), *Hydrogen and Serpentine: Their Role in the Localization of Gold Ores at the Oriental Mine*, Alleghany, California; Annual Meeting AIME; pre-print and full copy.

Diller, J.S. (1902 and 1910), *Topographic Development of the Klamath Mountains*; USGS Bull. 196 and 470-B.

Evans, David L. (1955), Miscellaneous materials in personal files.

Hershey, Oscar M. (1910), *Origin of Gold Pockets in Northern California*, MINING AND SCIENTIFIC PRESS, San Francisco, Dec. 3, 1910, pp 741-742.

Hotz, Preston E. (1964), *Nickeliferous Laterites in Southwestern Oregon and Northwestern California*, ECONOMIC GEOLOGY, Vol. 59, pp 355-395.

International Nickel, Boldt, Joseph—editor (1967), *The Winning of Nickel*.

Lindgren, Waldemar (1911), *Tertiary Gravels of the Sierra Nevada of California*; USGS Prof. Paper No. 73.

Landsweert, (1869), *Metallic Deposits-Gold Nuggets Theory as to Their Formation*; MINING AND SCIENTIFIC PRESS, San Francisco; Vol. 17, pp 82, 274, and 306.

LeConte, Joseph (1871), Excerpts from lecture in San Francisco, January 1871; as reported by MINING AND SCIENTIFIC PRESS, San Francisco.

Rucklidge, John (1972), *Chlorine in Partially Serpentinized Dunite*, ECONOMIC GEOLOGY, Vol. 67, pp. 38-40.

US Army Map Service (1958), Chico Regional Sheet, California. No. NJ 10.3.

White, D.E. (1968), *Environments of Generation of Some Base Metal Ore Deposits*, ECONOMIC GEOLOGY, Vol. 63, pp 301-335.

Economic Geology
Vol. 73, 1978, pp. 416-426

Basin Dynamics, Channel Processes, and Placer Formation: A Model Study

JOHN ADAMS, G. L. ZIMPFER, AND C. F. MCLANE

Abstract

The role of basin dynamics, tectonics, and channel processes in the formation of placers has not been well understood. In this study, these problems were investigated in a miniature drainage basin subjected to rejuvenation. Sediment samples collected during the experiment show the response of total sediment discharge and heavy mineral discharge to rejuvenation. Total sediment yield peaked shortly after rejuvenation began and then decayed exponentially. This decay does not characterize magnetite yield which was produced in discrete events separated by periods of little output.

Samples within the source area indicated that magnetite was concentrated and stored on uplands and hillslopes as well as in the floodplain. Valley fill contained horizontal layers deposited as the channel shifted laterally and trough-shaped scour and fill deposits, which marked former channel courses. Within these deposits magnetite was concentrated on bedrock channel bottoms, on alluvial channel bottoms ("false bottoms"), and in similar positions within terrace deposits.

Complex response dominates basin rejuvenation. Initially, placers are destroyed and transported from the basin. Aggradation then overloads channels and heavy minerals are stored. This is succeeded by degradation during which heavy minerals are concentrated and flushed from the basin. Superimposed upon this complex response are shorter periods of heavy mineral transport and storage controlled by internal geomorphic thresholds. In both cases, heavy mineral transport is directly controlled by channel activity. Storage occurs when the channel aggrades, and when the channel degrades heavy minerals are concentrated in the channels and transported from the basin.

Although basin rejuvenation destroys existing placers, it also creates an environment for the formation of new placers. In addition. the complex response of the source basin to a single uplift will cause the multiple reworking of alluvium necessary for placer formation.

Introduction

Two problems relating to placer formation have not yet been resolved. The first is the relationship of placer fields to basin dynamics and regional tectonics, and the second is the relationship of individual placers to channel activity.

Lindgren (1933, p. 225) and Bateman (1950, p. 229) suggest that placer fields are most readily formed in deeply weathered areas of moderate relief that are subjected to progressive small uplifts. These cause stream rejuvenation and successive reworking of stream deposits into rich placers. For the efficient reworking of sediment, a channel gradient must be poised between excessive degradation and aggradation, and Bateman (1950, p. 234) has suggested that a gradient of 0.006 (30 feet per mile) is optimal for placer formation. For similar reasons Sigov et al. (1972) conclude that placer deposits are not formed during periods of major uplift or subsidence because at these times the transport processes are not conducive to the reworking and concentration of low-grade deposits.

Despite the extensive description of bedrock and "false-bottom" placers in the literature, there is no generally accepted mode of formation of these deposits. Cheney and Patton (1967) consider that the concentration of heavy minerals at or near bedrock is axiomatic and that this concentration is caused by unusually large floods that rework the channel alluvium. By contrast Gunn (1968) suggests that the heavy minerals move downward as the gravels of the channel bed are agitated by streamflow. Tuck (1968) considers that heavy minerals are concentrated on the channel bedrock floor during periods of downcutting and that these are later buried by channel aggradation.

The explanation of Tuck is supported by a model study (Shepherd and Schumm, 1974) which indicates that magnetite forms a lag deposit on the channel floor when the channel degrades and that this concentration is buried when the channel later aggrades.

Two concepts recently developed by Schumm (1977) and his students can be used to explain channel and basin processes and relate these to placer

:ormation. The various components of a basin (floodplain, hillslopes, divides, tributaries, and main :hannel) respond at different times and at different :ates to external change (rejuvenation in this case) ınd this results in a complex response mechanism. The concept of complex response was first developed luring experimental work (Schumm and Parker, 1973). It has since been used to interpret valley ter-:aces along Douglas Creek in northwestern Colorado (Womack and Schumm, 1977). Schumm (1973, p. 307) illustrates four stages of complex response (Fig. 1) and describes complex response as follows:

> Incision occurred first at the mouth of the system, and then progressively upstream, successively rejuvenating tributaries and scouring the alluvium previously deposited in the valley. As erosion progressed upstream, the main channel became a conveyor of upstream sediment in increasing quantities, and the inevitable result was that aggradation occurred in the newly cut channel. However, as the tributaries eventually became adjusted to the new baselevels, sediment loads decreased, and a new phase of channel erosion occurred. Thus, initial channel incision and terrace formation was followed by deposition of an alluvial fill, channel braiding, and lateral erosion, and then, as the drainage system achieved stability, renewed incision formed a low alluvial terrace.

The second concept, geomorphic thresholds (Schumm, 1973), shows how channel pattern is controlled by slope thresholds within the fluvial system. 't has been known for some time that channel pattern s closely related to channel slope (Leopold and Wolnan, 1957). Recent work (Schumm and Khan, .972) has shown that a critical threshold slope exists :bove which a straight stream will become meanderng and a second critical threshold exists above which he meandering stream will become braided.

During the summer of 1976 research was conlucted at the Rainfall Erosion Facility (REF) at Colorado State University into the evolution of the lrainage net and the hydrology of an experimental !rainage basin. These studies followed the developnent of the basin from its initial configuration (two loping planes) to the formation of an "equilibrium" lrainage basin. This basin was then rejuvenated by . drop in base level, and the basin was then allowed o develop to equilibrium. Morphological similarities etween the REF drainage pattern, drainage density, .nd stream profile to those of natural drainage basins ıave been shown by earlier workers (Parker, 1977). Magnetite had been observed to concentrate in the REF channels during an earlier study (Macke, 977) in an aggrading river channel, but the present :periment provided an opportunity to study placer ormation in a drainage basin as it underwent reuvenation.

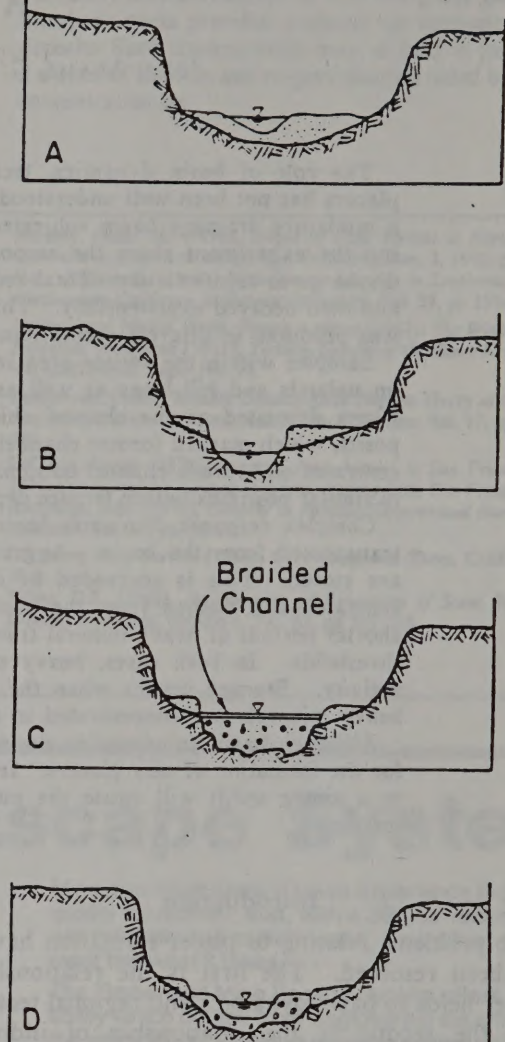


FIG. 1. Diagrammatic cross sections of experimental channel 1.5 m from outlet of drainage system (base level) showing response of channel to one lowering of base level (from Schumm, 1973).

A. Valley and alluvium, which was deposited during previous run, before base-level lowering. The low width-depth ratio channel flows on alluvium.

B. After base-level lowering of 10 cm, channel incises into alluvium and bedrock floor of valley to form a terrace. Following incision, bank erosion widens channel and partially destroys terrace.

C. An inset alluvial fill is deposited as the sediment discharge from upstream increases. The high width-depth ratio channel is braided and unstable.

D. A second terrace is formed as the channel incises slightly and assumes a low width-depth ratio in response to reduced sediment load. With time, in nature, channel migration will destroy part of the lower terrace and a flood plain will form at a lower level.

The time span of tectonic processes makes it impossible to observe the effect of these changes on placer formation in any natural basin. However, the drop in base level in the REF simulates basin rejuvenation and from a study of heavy mineral pro-

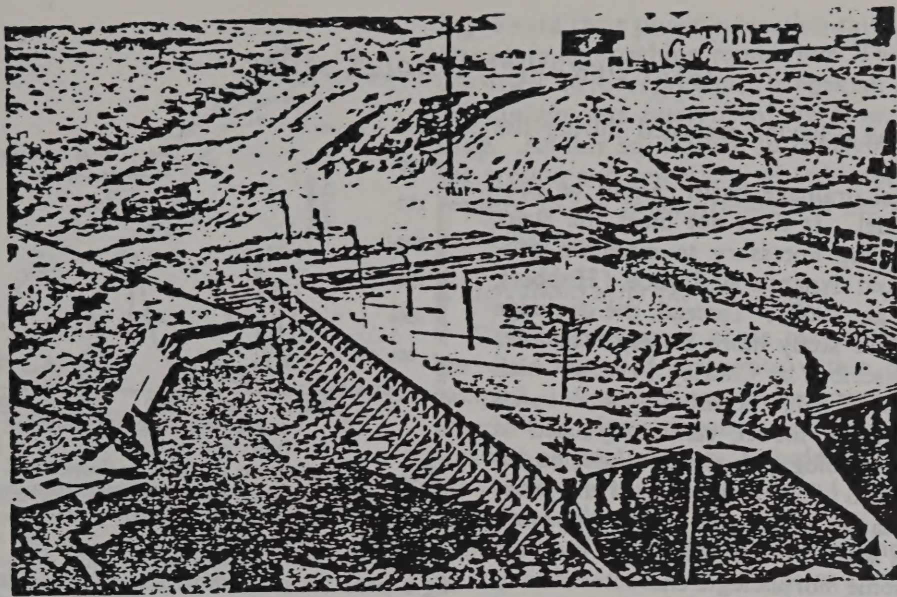


FIG. 2. The Rainfall Erosion Facility (REF) before enclosure. Crane shows scale, box is 15 m long.

duction and storage in the basin during this period it is possible to relate placer formation to the rejuvenation processes.

The quantity of heavy minerals stored in the REF allows the formation of placer deposits to be related to the complex response of the basin to rejuvenation. This estimation together with observations of channel activity also allows the recognition of conditions favorable to placer formation. Thus it is possible to use the REF to relate basin and channel processes to the formation of placers and to study the heavy mineral export from a drainage basin through time.

Equipment and Methods

The Rainfall Erosion Facility consists of a 9.1 m by 15.2 m plywood box (Fig. 2) 1.8 m deep that was filled to an average depth of 0.9 m with a 1:1 mixture of local topsoil and plaster sand (mean size 1.55 phi, standard deviation 1.9 phi) which contained 0.4 percent magnetite. This material was compacted by rolling and is referred to as bedrock in contrast with the reworked sediments (alluvium) found in the channels.

Controlled rainfall was applied to the surface at rates of 20, 28, and 56 mm/hr. The highest intensity of rainfall was used to develop the drainage basin and for most of the hydrologic investigations. This resulted in a runoff on the order of 1.3 liters per second. A small flume was attached to the front of the REF to control base level and provide an outlet for runoff and sediment. Sediment production from the model basin was measured only at this outlet flume, and thus information on sediment supply from the individual tributaries to the main channel is not available. During the experiment base level was lowered by removing a plank from beneath the outlet flume and lowering the flume.

During the development of the basin, sediment samples were taken at 15-minute intervals (samples 11–18 and 20–28) for much of the run, and at 30-minute intervals (samples 31–46) for the remainder. Sample intervals during other times (samples 1–10, 19, 20, 29, 30, and 47–50) varied from 8 minutes to 134 minutes with a mean of 31 minutes. Each sample represents the *total* water and sediment discharge from the basin for a measured period that varied between 1.5 and 2.8 seconds. These data were later corrected to a constant sampling period of 2 seconds and then adjusted so that each sample was equivalent to 2 seconds of the 56 mm/hr rain intensity, so that the data would not be biased by the sampling period.

The sediment samples were suction filtered, dried, and weighed to determine the sediment yield. To remove clay and low-density components these samples were washed and panned to about one-eighth of their initial volume in a gold pan. The fine grain size of the magnetite (mean size 3 phi) precluded removal of the fine grains by wet sieving since about one-quarter of the magnetite would also have passed through the sieve. The cleaned samples were subsequently dried and the magnetite fraction separated by a hand magnet and weighed. The error involved in this procedure was approximately 6 percent based on replicate sampling. Magnetite yields were then compared to the total sediment yield, and the deviations from the expected values related to the observations

of basin and channel behavior recorded during the experiment.

This experimental drainage basin cannot be considered a scale model. It does not meet the strict rules of scale modeling necessary to maintain geometric, kinematic, and dynamic similitude (Hubbert, 1937; Albertson et al., 1960). Instead, the drainage basin described in this study more closely resembles the analog model described by Chorley (1967) which Hooke (1968, p. 393) calls "a small system in its own right." An analog model, commonly used in experimental geomorphology, does not model an individual drainage basin but models some of the properties of drainage basins in general. This approach is valid if the gross scaling relationships are met, the model reproduces some morphologic characteristics of the prototype (a natural system), and the process which produced the morphologic characteristics in the laboratory can logically be assumed to have the same effect on the prototype (Hooke, 1968). Hooke also demonstrates that a model of this type is necessary because the scale modeling of alluvial systems is not possible at the present state of knowledge. The failure to maintain scaling ratios prevents the direct extrapolation from the model to the prototype, but we consider that the results of experiments of this type may be applied qualitatively to the field, even though the rules of scale modeling were not observed.

Results

The weight of magnetite in the undisturbed material was determined to be 0.408 percent with a standard deviation of 0.025 percent. The magnetite concentration in the sediment discharge samples ranged from 0.068 to 2.01 percent, with a mean value of 0.37 percent. When the magnetite concentration in the samples was plotted against cumulative precipitation (an index of time or stage), the samples showed several distinct peaks (Fig. 3).

Immediately following the base-level drop, sediment yield increased sharply (Fig. 4). After reaching a peak the data show exponential decay similar to the sediment yield curve described by Parker (1977, see also Schumm, 1977) based on more detailed sampling. The sudden lowering of base level produced a knickpoint which migrated up the main channel. As this knickpoint passed tributary junctions, knickpoints formed in the tributaries. These migrating knickpoints defined a "wave of dissection" (Howard, 1971, p. 30) that moved through the drainage system and was one cause of changes in the amount of sediment released to the main channel and the fluctuations superimposed on the general sediment decay curve.

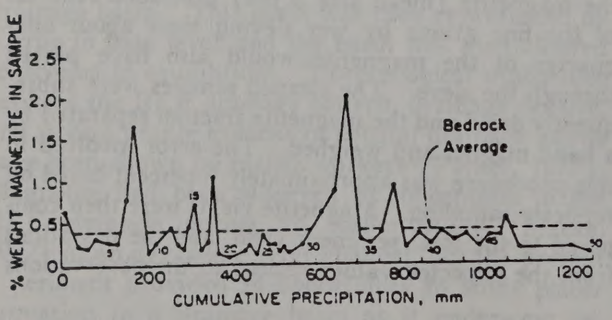


FIG. 3. Variation in magnetite concentration in the samples with cumulative rainfall. The horizontal line represents the average magnetite concentration in the bedrock.

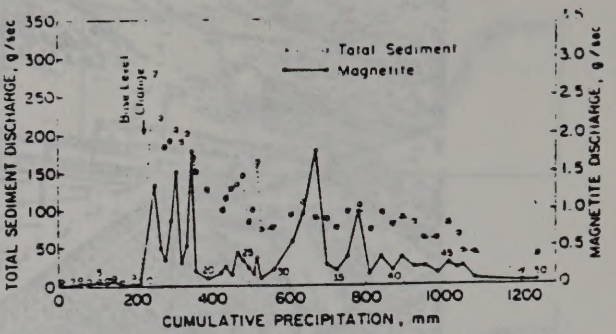


FIG. 4. Production of total sediment and magnetite from the basin. Data have been corrected to a constant sampling period of 2 sec of 56 mm rainfall.

In contrast with the sediment production, exponential decay was not shown by the magnetite production. Instead, magnetite was produced during a series of discrete events that were separated by periods during which little magnetite was moved out of the basin. The data represented by Figure 4 indicate that both sediment and magnetite leave the basin as pulses but that the magnitude of the pulses (relative to the expected values) is relatively greater for the magnetite than for the total sediment production.

The data from Figure 4 were used to plot the cumulative production of magnetite against the cumulative production of total sediment (Fig. 5). The samples indicate that a total of 7,280 kg of sediment and 27.25 kg of magnetite was moved out of the basin during the experiment. Figure 5 shows that there are departures from the straight line of slope 0.00408 that would be expected if the magnetite had been transported from the basin at the same rate as the total sediment.

At three points (samples 11, 15, and 18) slightly more magnetite was removed from the drainage basin than can be explained from the yield of total sediment. This excess magnetite may have been derived from the reworking of alluvial deposits inherited from the preceding cycle of development.

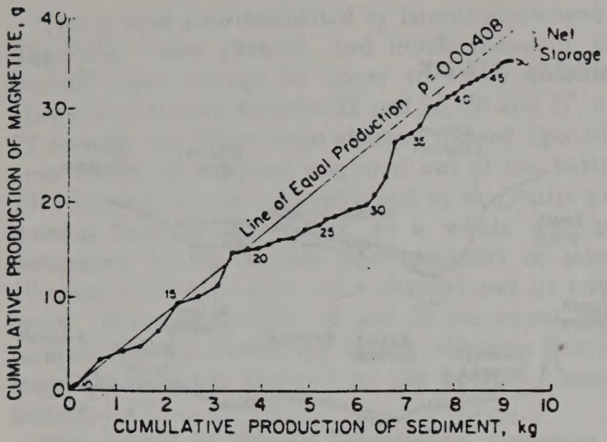


FIG. 5. Cumulative plot of magnetite vs total sediment in the samples. Storage at any point can be calculated from the deviation between the data points and the line of equal production. Points above the line represent transport of magnetite in excess of magnetite freed from bedrock.

An indication of the magnetite stored in the basin can be calculated from the difference between the observed and expected values for each point on the cumulative curve (that is, the vertical deviations of the sample points from the line of equal production). An increase in magnetite storage represents the formation of placers within the basin. By contrast, a decrease in storage represents the flushing of magnetite from the basin and the potential formation of placers downstream of the basin.

The plot of magnetite storage against cumulative rainfall (Fig. 6) shows that magnetite storage in the basin changed dramatically during the experiment. As the end of the experiment was approached the rate of magnetite storage decreased. Figure 4 indicates that sediment and magnetite discharge were decreasing toward their pre-rejuvenation rates and the basin was again approaching equilibrium. Thus the samples discussed here represent changes in the basin as equilibrium was interrupted by a major base-level drop. As this adjustment to rejuvenation occurred, the erosion and reworking of sediment in the basin resulted in the concentration and net storage of 2.5 kg of magnetite.

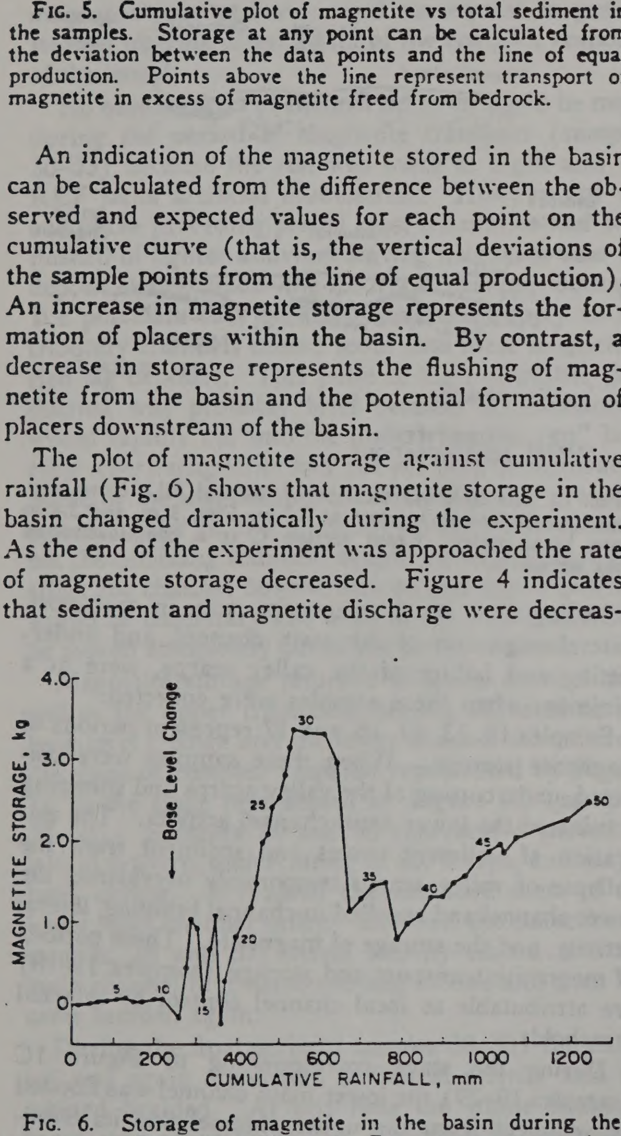


FIG. 6. Storage of magnetite in the basin during the experiment. Data from Figure 5. Zero storage assumed at start of data collection. negative storage represents removal of existing magnetite placers.

Observations after the experiment

After completion of phase two of the experiment, the distribution of magnetite within the basin was sampled. Magnetite was concentrated in the channel alluvium as placer deposits, as indicated below, and was also concentrated on the bedrock surface between the channels where rainsplash had removed much of the lighter alluvium.

Samples of the drainage basin surface were taken at 24 grid locations. At each point a small (20–40 g) sample was scraped from the surface and analyzed for the proportion of weight of magnetite in the sample. These weights ranged from 0.91 to 4.88 percent of the sample weight, with a mean of 1.58 percent. The samples can be divided into three categories: upland samples (13 samples), hillslope samples (4 samples), and floodplain samples (7 samples). Mean proportions of magnetite for each category were calculated. Upland and hillslope concentrations were found to be approximately the same, with means of 1.13 percent and 1.05 percent, respectively. Floodplain samples had a mean of 2.72 percent, more than twice that found in hillslope and upland samples.

The amount of magnetite stored on upland and hillslope surfaces in the basin can be estimated from the product of the drainage area (86.5% of the total basin area of 105 m^2 is upland and hillslope), sample depth (no greater than 0.006 m), bulk density of the surficial material (determined to be 1,390 kg/m^3), and the proportion of magnetite in the samples (1.1%).

The calculation indicates that approximately 8.3 kg of magnetite was held essentially in situ on the bedrock surface. This is an overestimate of the actual magnetite stored because both the actual sampling depth and the thickness of the magnetite surface layer were less than 0.006 m.

Transverse cross sections of the channel were excavated to study the alluvial fill. The valley fill (Fig. 7) contained horizontal layers, deposited as the channel shifted laterally, and trough-shaped scour and fill deposits, which marked former channel courses. Magnetite was frequently concentrated on bedrock channel bottoms, alluvial channel bottoms (false bottoms), and on the bedrock interface of terrace deposits. It was commonly associated with

coarser grained alluvium of presumed hydraulic equivalence. A bulk sample of one of these placer deposits contained 15 percent magnetite (38 times background), and locally concentrations of pure magnetite were observed. Eleven nonrandom samples were taken to represent these placer deposits.

These representative samples were combined with the seven grid samples to give a mean concentration of magnetite in the channel fill and terrace deposits of 5.34 percent (13 times background). Valley deposits comprise 13.5 percent (14.11 m^2) of the basin area and have a bulk density of 1,340 kg/m^2. Assuming a mean sampling depth of 0.006 m, this indicates that 6.1 kg of magnetite was stored in these deposits, and, hence, channel storage accounts for slightly less than half of the total basin storage.

The combined estimates indicate 14.4 kg of magnetite were stored in the basin. This is an overestimate of the actual storage because upland sample depths have been overestimated and channel concentrations may be biased toward higher values.

Analysis of the sediment discharge samples suggests a total storage of 4.5 kg, which is in fair agreement with the magnetite estimated by sampling the basin after the experiment.

Magnetite storage and basin activity

The equilibrium of the lower main channel before the base-level drop corresponds to Figure 1A. Little sediment was being transported from the basin (samples 1–10, Figure 4), and both total sediment and magnetite discharge were approximately constant. Very short periods of aggradation and degradation controlled by intrinsic thresholds were observed in the lower main channel, but these had little effect on sediment discharge.

Initial channel response to base-level drop corresponds to Figure 1B. Twenty-seven minutes after sample 10 was collected, base level was lowered 203 mm to rejuvenate the basin. Immediately, the main channel narrowed and downcut through existing alluvial deposits. Total sediment production increased rapidly but sediment export from the basin was not continuous, as shown by samples 11 to 18, and magnetite storage (Fig. 6) went through a period of rapid fluctuation.

Observations made during this period provide an explanation for these fluctuations. Samples 11, 15, and 18 were collected during periods of downcutting. Sample 11 was collected as the first major knickpoint migrated up the main channel. Just before sample 15 was collected, a smaller knickpoint migrated up the main channel. Between samples 17 and 18 the lower main channel narrowed and incised. Magnetite was deposited along the banks of the incised channel, indicating that much magnetite was in transport. Lateral migration of the main channel, and undercutting and failure of the valley scarps, were at a minimum when these samples were collected.

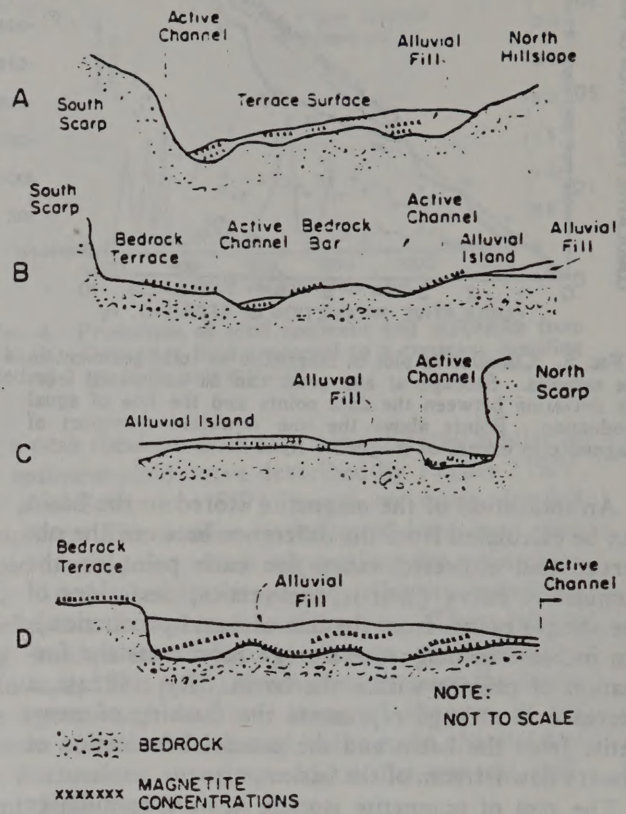


FIG. 7. Sketch cross sections of channel fill. Cross sections (top to bottom) taken at 7.7 m, 5 m, 5 m, and 2.4 m above basin outlet. Cross section C is a continuation of cross section B.

Samples 12, 13, 14, 16, and 17 represent periods of magnetite storage. When these samples were collected, undercutting of the valley scarps and slumping dominated the lower main channel activity. The migration of sediment waves and sediment from the collapse of valley scarps temporarily overloaded the lower channel and resulted in channel braiding, lateral activity, and the storage of magnetite. These periods of magnetite transport and storage (samples 11–18) are attributable to local channel control by internal thresholds.

During the stage corresponding to Figure 1C (samples 19–29) the lower main channel was flooded by sediment from the upper channel and its tributaries. Before the collection of sample 23 response to base-level drop was confined mainly to the lower main channels and the lower parts of its tributaries. After sample 23 was collected, the upper basin responded actively and the lower 5 m of the basin played a more passive role.

This period was dominated by lateral movement in the lower main channel, and much sediment was flushed from storage in upper tributary channels. However, between samples 22 and 23, 26 and 27, and 27 and 28, the lower main channel incised (possibly after waves of sediment migrated out of the basin). This incision was not accompanied by magnetite production because the system as a whole was still dominated by the erosion and transport of stored alluvium from the upper main channel and its tributaries. Between samples 18 and 28 the upper basin was completely flushed of lighter alluvium leaving magnetite deposits exposed on the bedrock channel bottom.

This period of channel aggradation, transport of magnetite-poor sediment, and consequent magnetite storage was the direct result of the complex response of the basin.

No observations of channel behavior could be made during the period of magnetite transport (samples 30–33) because the run was made at night and the REF lacks artificial illumination. However, at the end of the preceding period most tributaries had been flushed of lighter alluvium leaving magnetite as a lag deposit lying upon the bedrock channel bottoms. We postulate that declining sediment supply to the tributary channels caused scour into these magnetite-rich lag deposits. This pulse of high magnitite production was probably brief because these channels would rapidly cut into the underlying "barren" bedrock where magnetite had not yet been concentrated. Some of the magnetite from this pulse was probably stored briefly in the alluvium of the main channel, but the declining sediment supply would progressively affect the channel downstream so that any deposition would be temporary and most of the sediment would be moved completely out of the lower channel.

Magnetite storage returned to lower channel control as the upper tributaries became adjusted to the base-level change and the lower channel shifted from the stage of complex response represented by Figure 1C to the stage represented by Figure 1D (samples 34–50). Samples 34 and 35 represent a period of lateral activity, undercutting of the scarps, and major bank caving. A braided channel filled most of the lower main channel valley. Between the collection of samples 36 and 37, lateral activity ceased and the main channel first narrowed and incised and then became braided again.

During the final period (samples 38–50) the channel was again dominated by lateral activity and remained braided. At this time the whole drainage basin was adjusted to the rejuvenation, and magnetite storage was again controlled by local overloading of the channels.

The formation of bedrock placer deposits and false-bottom placers in particular can be explained by the concepts described above (Schumm, 1977). Channel behavior is strongly influenced by the response of the drainage basin to external change (in this case rejuvenation) and to the crossing of internal thresholds. During periods of low sediment supply the channel can scour and winnow the channel fill to bedrock and so concentrate any heavy minerals present in the channel fill. These concentrations may be on the bedrock surface or at an intermediate depth in the alluvium. Less extreme scour, as expected from the concepts of complex response and geomorphic thresholds, would imply only partial scour of the channel fill and the formation of false-bottom placers above the bedrock surface (e.g., Fig. 1D). Assuming some preexisting distribution of heavy minerals within the channel fill, the deeper the scour, and the more sediment reworked, the richer the deposit concentrated on the scour bottom. Hence, the concentration of heavy minerals in a channel-fill deposit will generally increase downward to the bedrock, as observed in the field examples.

The interpretive framework presented here shows that channel response to a single rejuvenation event results in a period of complex response. This complex response and the adjustment of the channel to internal thresholds result in the formation of multistory placers. Similar multistory placers have been observed during channel aggradation following a rise in base level (Macke, 1977). A degrading channel is more favorable to the formation of rich placers, though the concepts of complex response and geomorphic thresholds ensure that deposition in an aggrading channel will be interrupted by periods of degradation and the formation of false-bottom placers. This may be the origin of placer deposits that occur high above the bedrock surface, such as those in the Ross gold field.

Discussion

Placer morphology

In the present work, similarities between REF heavy mineral concentrations and natural placers were observed. For example, residual placers formed on the model uplands are comparable to the dispersed residual deposits that Krook (1968) considers might be held in situ by vegetation and later remobilized by a climatic change to enter the channels as a workable deposit. Four other placer types were also observed.

Active channel placers: During pauses in the experiment when the channel was drained while it was degrading, magnetite concentrations were observed to form elongate pay streaks on the bedrock. During aggradation these deposits were covered by alluvium

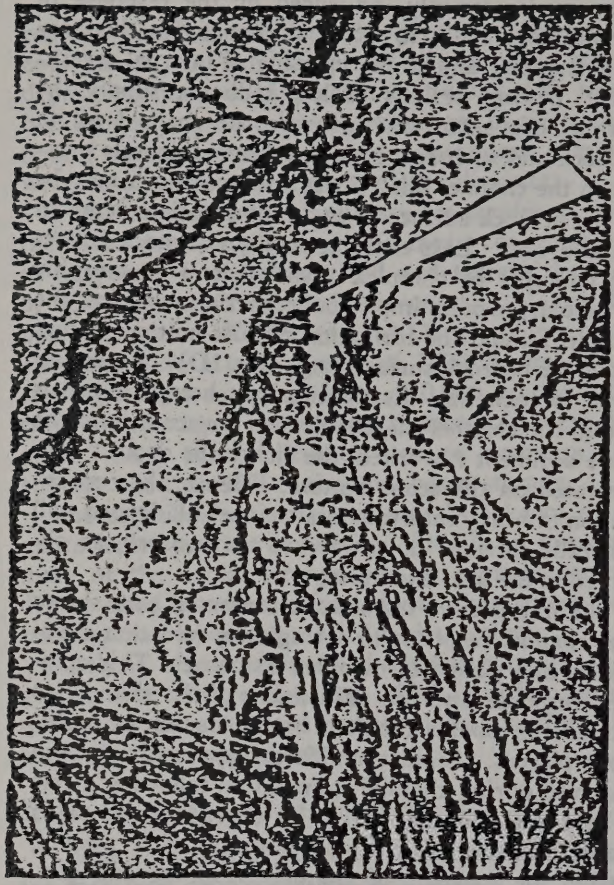


FIG. 8. Magnetite concentrations in the active lower main channel with channel drained for mapping. Flow is toward viewer; grid spacing 1 m. Arrow indicates regularly spaced scours.

to form buried concentrations, which were observed in the channel cross sections.

Figure 8 shows several features of areal heavy mineral concentrations in the model channel. The photograph shows bedrock terraces with scant coarse alluvial cover (left center), rhomboidal magnetite streaks are present at bottom center, and a series of regularly spaced scours and associated magnetite concentrations are present at top center. The regular pattern of scours was caused by the standing waves in the channel that resulted from the shallow water depths and steep slopes in the model. Thus they, like those of Wertz (1949), are probably confined to model studies and should not be considered to represent field examples.

Scour hollows occur at the confluence of tributaries in the model basin (Fig. 9). Mosley and Schumm (1977) suggest that these scours may be a preferred location for placer deposits because scours are deepest at the confluence, and the position of the scour remains in approximately the same location. This promotes the reworking of sediment and concentration of heavy minerals. Their work is supported by the occurrence of magnetite in the scours at a few confluences in the present work (Fig. 9). The presence of scour hollows does not in itself ensure the accumulation of heavy minerals, because the turbulence generated may be sufficient to move both sediment and heavy minerals.

Channel bedrock placers: Almost all channel fills examined in the model basin showed magnetite concentrations on the bedrock-alluvium contact (Fig. 7). These deposits ranged from a thin veneer on bedrock to comparatively thick deposits that formed in the thalweg.

Channel bedrock concentrations have been extensively worked and are the most widely cited in the literature. In the La Cholla area of Arizona, gold was found at 13 and 5 m above bedrock, but "the richest gold-bearing gravel occurs within a few inches of the bedrock" (Wilson, 1961, p. 30). In a second Arizona example, Schrader (1915) described placers in the Greaterville area in which the pay streaks were confined to narrow channels 2 to 4 m wide on bedrock that widened and became more deeply buried

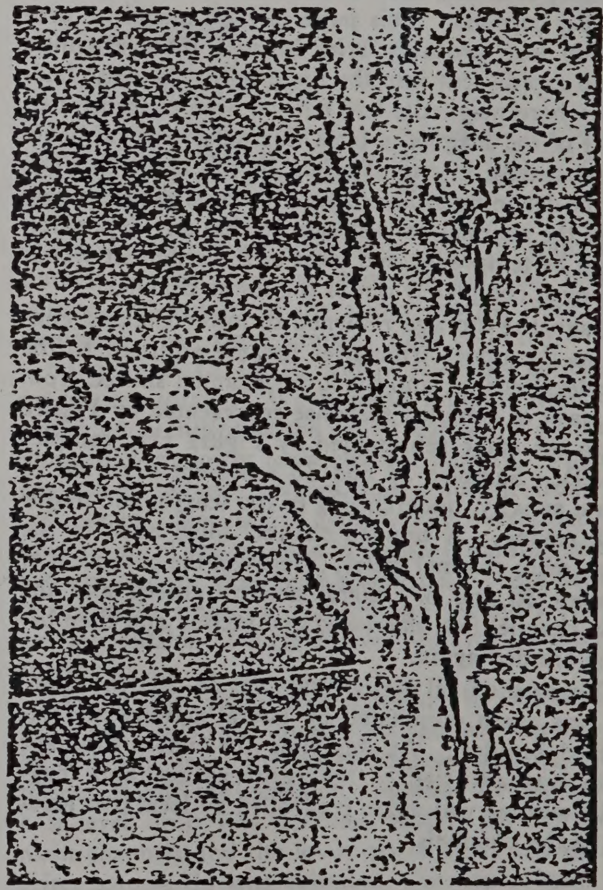


FIG. 9. Scour and associated heavy mineral concentration at tributary confluence.

downstream. Cheney and Patton (1967) conclude from field evidence that the concentration of heavy minerals on or near the bedrock surface is axiomatic.

False-bottom placers: One feature observed in both this experiment and in studies by Macke (Schumm, 1977, fig. 7-32) is the presence of magnetite concentrations within the channel fill. These have been widely recognized in natural deposits as false-bottom placers. Two placer deposits above the bedrock concentration were recognized in the Excelsior claim in the Sierra Nevada by Lindgren (1911, fig. 15) at 8 and 18 m above the bedrock. Eight payable levels separated by barren gravels were present in a 120-meter shaft that failed to reach bedrock in the Ross gold field of New Zealand (Acheson, p. 119, *in* Galvin, 1906). These false-bottom placers may interrupt the rapid increase in value as the bedrock is approached, as is shown in a Yukon terrace gravel (Koch and Link, 1971, fig. 15.2).

Terrace placers: During response to the base-level drop the main channel formed many bedrock terraces that were later abandoned when the channel downcut. These terraces were covered by alluvium and contained heavy mineral concentrations similar to those described from the channel fill deposits. These concentrations indicate that multiple terrace levels may have been formed by a single base-level drop. These terraces and placers correspond to the bench or terrace deposits worked in many gold fields such as in the Sierra Nevada of California (gravels to 900 m above stream level) described by Lindgren (1911) and at Nome, Alaska (200 m above stream level; Lindgren, 1933, p. 230).

Process model for placer formation

The observations and results presented in this paper indicate that placer formation can be dominated by processes occurring in the fluvial system (Schumm, 1977). These processes may be classified into basin processes (dominated by complex response) and channel processes (dominated by geomorphic thresholds).

Complex response characterizes the rejuvenation of the experimental drainage basin. Initially, the lower main channel downcuts while the upper basin remains unaffected by the drop in base level. When knickpoints migrate into the upper basin, the lower main channel is flooded with excess sediment and aggrades. Later, when the upper basin comes into equilibrium with the new base level, the lower main channel will again downcut until it is also in equilibrium. As this response occurs, magnetite is concentrated in the channels and transported from the drainage basin. In general, when the lower main channel is degrading, magnetite is in transport; when aggrading, the magnetite is stored in the alluvial fill.

Superimposed upon this sequence of magnetite transport and storage are shorter periods of transport and storage that are controlled locally by internal thresholds. Channel behavior in the lower main channel can be divided into periods dominated by lateral cutting and periods dominated by vertical downcutting. These alternating periods can be explained by the concept of intrinsic thresholds (Schumm, 1973). During periods of lateral activity the channel is braided or meandering (sinuous), is aggrading, and magnetite is being stored in the drainage basin. During periods dominated by downcutting the channel is straight or meandering (less sinuous), is degrading, and magnetite is being exported from the drainage basin.

Three sources (response to rejuvenation, normal bedload transport, and valley scarp collapse) cause fluctuation in the sediment transport rate, control channel aggradation and degradation, and determine channel slope, pattern, and the type of channel activity. Channel activity, in turn, controls the storage or transport of magnetite from the experimental basin and, hence, the formation or destruction of placers.

The model presented here suggests that the complex response to a single uplift will supply the multifold reworking of channel alluvium that is required for placer formation. Thus, although basin rejuvenation may destroy existing placers, it creates a suitable environment for the formation of new placers.

The conceptual model may be used to explain placer deposits related to the Southern Alps of New Zealand. Despite rapid uplift and modern erosion of gold-bearing schists, few of the short, steep rivers of South Westland carry workable placer deposits. Where present, these placers consist of small pockets of coarse "shotty" gold. Here channel slopes are too steep, and uplift too rapid, to allow the accumulation of much gold in the active channel.

Rapid deposition of the erosion products occurs in the fiords left by the glacial retreat, but again no workable deposits have been found in these sediments. Economic deposits are found only where sediments have been reworked into marine placers, for example, at Okarito and Gillespie Beach.

The older sediments of the Ross gold field were deposited in a similar environment, but here continuous deposition was interrupted by periodic channel degradation and economic multistory placers were formed. These false-bottom placers, which are at least 120 m above bedrock, are most easily explained by the concepts developed here.

Basin rejuvenation caused by uplift is similar to that caused by a drop in base level (as in the model). therefore, channels in coastal areas cut during glacial

periods of low sea level might contain higher concentrations of heavy minerals than alluvium, which filled the channels during interglacial periods. Such submerged fluvial deposits have been worked for tin ore in southeastern Asia, and some Alaskan coastal placers may also be of a similar origin.

Conclusions

We hope that the results presented here will be of general interest to geologists and others involved in the understanding of fluvial placers. It should be stressed that the results of the model experiment must be interpreted qualitatively, and that some processes in the REF may not be comparable to natural situations. This experiment has considered only a degrading, rejuvenated basin and has not studied the associated depositional environment. The deposition of heavy minerals in an alluvial-fan environment was the subject of earlier experimental work (Macke, 1977) in the REF.

In this paper both field and model evidence have been used to establish a proposed interpretive framework (conceptual model) for the formation of placers. We lack the opportunity to test the model ourselves, and therefore it is our hope that this framework will be criticized, discussed, and tested.

We conclude that:

1. The REF provides an acceptable analog model for placer formation in a natural system; in both the REF and the natural system deposit morphology is similar.

2. Complex response and the crossing of internal geomorphic thresholds can explain REF channel behavior following basin rejuvenation.

3. Total sediment discharge after basin rejuvenation reaches a peak and then follows an exponential decay curve with superimposed fluctuations. Magnetite discharge does not follow a decay curve but is released as a series of pulses.

4. Only a portion of the magnetite released during the rejuvenation remained in the basin, but much of this was concentrated in the channel deposits.

5. Basin rejuvenation may initially destroy existing placers, but conditions following rejuvenation are favorable to the formation of new placers.

6. Heavy mineral transport and storage and, thus, placer formation can be related to channel behavior.

7. Placers form during alternate periods of mild aggradation and degradation that result from the exceeding of the internal geomorphic thresholds which may follow rejuvenation.

8. Multistory placers with valuable mineral concentrations are favored by degrading channels but may also form in aggrading systems. Multistory placers are formed as the result of complex response and the crossing of internal geomorphic thresholds.

Acknowledgments

We wish to thank Drs. Schumm, Ethridge, and Thompson, and members of the 1976 geomorphology seminar for helpful comments on an earlier draft. The work was supported by a grant from the U. S. Army Research Office and was completed while Adams was supported at Colorado State University by a scholarship from the New Zealand University Grants Committee.

J. A.
VICTORIA UNIVERSITY
WELLINGTON, NEW ZEALAND
G. L. Z. AND C. F. McL.
DEPARTMENT OF EARTH RESOURCES
COLORADO STATE UNIVERSITY
FORT COLLINS, COLORADO 80523
March 30, August 30, 1977

REFERENCES

Albertson, M. L., Barton, J. R., and Simons, D. B., 1960, Fluid mechanics for engineers: Englewood Cliffs, New Jersey, Prentice-Hall, 562 p.
Bateman, A. M., 1950, Economic mineral deposits, 2nd ed.: New York, John Wiley and Sons, 916 p.
Cheney, E. S., and Patton, T. C., 1967, Origin of the bedrock values of placer deposits: ECON. GEOL., v. 62, p. 852–853.
Chorley, R. J., 1967, Models in geography, chap. 3, *in* Chorley, R. J., and Hagget, P., eds., Models in geography: London, Methuen, 816 p.
Galvin, P., ed., 1906, The New Zealand mining handbook: Wellington, New Zealand Government Printer, 589 p.
Gunn, C. B., 1968, Origin of the bedrock values of placer deposits [disc.]: ECON. GEOL., v. 63, p. 86.
Hooke, R. L., 1968, Model geology: Prototype and laboratory streams [disc.]: Geol. Soc. America Bull., v. 79, p. 391–394.
Howard, A. D., 1971, Stimulation of stream networks by headward growth and branching: Geog. Analysis, v. 3, p. 29–50.
Hubbert, M. K., 1937, Theory of scale models as applied to the study of geologic structures: Geol. Soc. America Bull., v. 48, p. 1459–1520.
Koch, G. S., and Link, R. F., 1971, Statistical analysis of geological data, vol. 2: New York, John Wiley and Sons, 438 p.
Krook, Leendert, 1968, Origin of bedrock values of placer deposits [disc.]: ECON. GEOL., v. 63, p. 844–846.
Leopold, L. B., and Wolman, M. G., 1957, River channel patterns: braided, meandering, and straight: U. S. Geol. Survey Prof. Paper 282-B, 85 p.
Lindgren, Waldemar, 1911, The tertiary gravels of the Sierra Nevada of California: U. S. Geol. Survey Prof. Paper 73, 226 p.
—— 1933, Mineral deposits, 4th ed.: New York, McGraw-Hill, 930 p.
Macke, D. L., 1977, Stratigraphy and sedimentology of experimental alluvial fans: Unpub. M.S. thesis, Colorado State Univ.
Mosley, M. P., and Schumm, S. A., 1977, Stream junctions—a probable location for bedrock placers: ECON. GEOL., v. 72, p. 691–694.
Parker, R. S., 1977, Experimental study of basin evolution and its hydrologic implications: Unpub. Ph.D. dissert., Colorado State Univ.

Schrader, F. C., 1915, Mineral deposits of the Santa Rita and Patagonia Mountains, Arizona: U. S. Geol. Survey Bull. 582, 373 p.

Schumm, S. A., 1973, Geomorphic thresholds and complex response of drainage systems, *in* Morisawa, Marie, ed., Fluvial geomorphology: SUNY, Binghamton, N.Y., Publications in Geomorphology, p. 299–310.

—— 1977, The fluvial system: New York, Wiley Interscience, 352 p.

Schumm, S. A., and Khan, H. R., 1972, Experimental study of channel patterns: Geol. Soc. America Bull., v. 83, p. 1755–1770.

Schumm, S. A., and Parker, R. S., 1973, Implications of complex response of drainage systems for Quaternary alluvial stratigraphy: Nature, Phys. Science, v. 243, p. 99–100.

Shepherd, R. G., and Schumm, S. A., 1974, Experimental study of river incision: Geol. Soc. America Bull., v. 85, p. 257–268.

Sigov, A. P., Lomayer, A. V., Sigov, A. V., Storozhenko, K. Y., Khrypor, V. N., and Shub, I. F., 1972, Placers of the Urals, their formation, distribution, and elements of geomorphic prediction: Soviet Geography, v. 13, p. 375–387.

Tuck, Ralph, 1968, Origin of bedrock values of placer deposits [disc.]: Econ. Geol., v. 63, p. 191–193.

Wertz, J. B., 1949, Logarithmic pattern in river placer deposits: Econ. Geol., v. 44, p. 193–209.

Wilson, E. D., 1961, Gold placers and placering in Arizona: Arizona Bur. Mines Bull. 168, 124 p.

Womack, W. R., and Schumm, S. A., 1977, Terraces of Douglas Creek, northwestern Colorado: An example of episodic erosion: Geology, v. 5, p. 72–76.

THIS PAGE
INTENTIONALLY
BLANK